

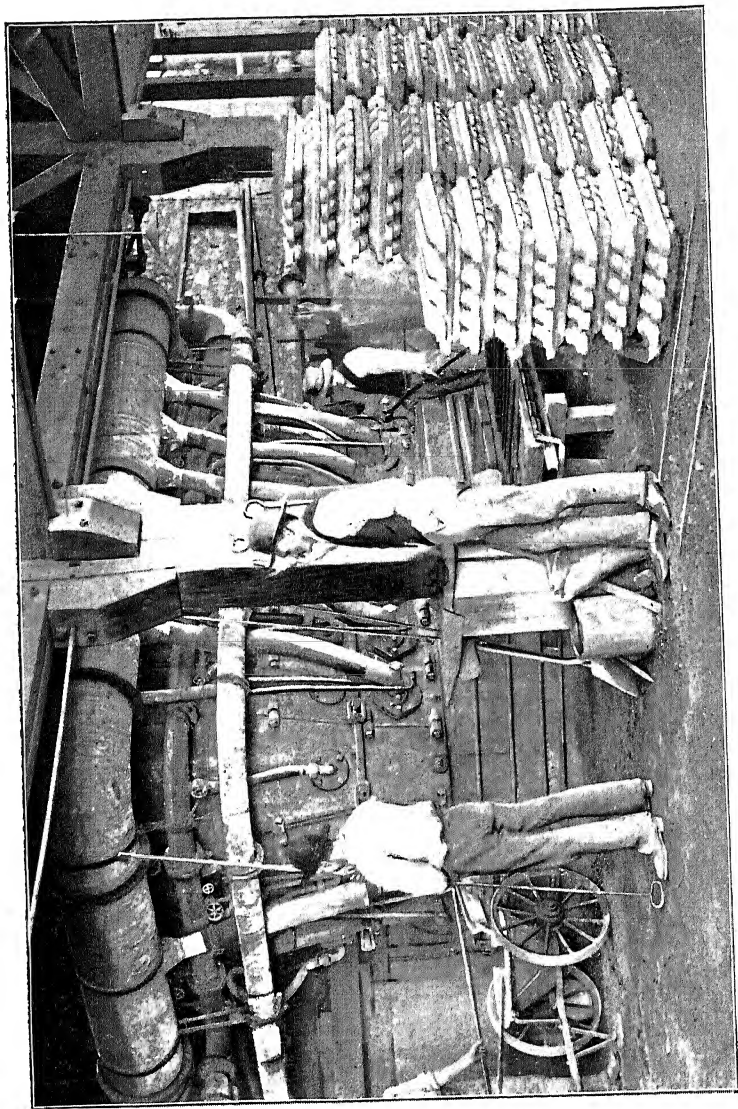
THE  
METALLURGY OF LEAD.

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WATER-JACKETED SMELTERS OF BRITISH BROKEN HILL CO.

# THE METALLURGY OF LEAD.

BY

HENRY F. COLLINS,

ASSOCIATE ROYAL SCHOOL OF MINES,  
ASSOCIATE-MEMBER OF THE INSTITUTION OF CIVIL ENGINEERS, MEMBER OF THE INSTITUTION  
OF MINING AND METALLURGY, MEMBER OF THE AMERICAN INSTITUTE OF MINING  
ENGINEERS, OF THE MINERALOGICAL SOCIETY OF GREAT  
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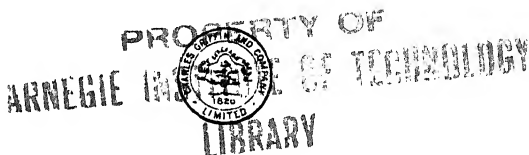
*Being one of a Series of Treatises on Metallurgy written by  
Associates of the Royal School of Mines.*

EDITED BY

SIR W. C. ROBERTS-AUSTEN, K.C.B., D.C.L., F.R.S.

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1910.

## Dedication.

---

*TO MY FATHER,*

J. H COLLINS, F.G.S.,

PAST-PRESIDENT OF THE INSTITUTION OF MINING AND METALLURGY,

*&c., &c., &c.*

I am indebted for an early training in those branches of Natural and Applied Science, of which he has been an earnest student and an accomplished teacher. For this early training I can never be sufficiently grateful, and to it must in justice be attributed such measure of success as I may hereafter achieve. To my father, therefore, this book is (without permission) dedicated by

THE AUTHOR.

## PREFACE TO THE SECOND EDITION.

---

THE progress made in the Metallurgy of Lead since the appearance of the First Edition has been so rapid that, in the endeavour to bring the work up-to-date, not only has it been necessary to add three entirely new Chapters, but two-thirds of the remaining Chapters have been re-written throughout and considerable additions have been made to the others, the nett result being to practically double the size of the book. Special attention may be directed to the new Chapters on the Blast-roasting of Lead Ores and on Flotation processes, also to the completely re-written important Section of Six Chapters on Blast-furnace Smelting, and to the Chapters on Roasting, on Flue-dust, and on the Treatment of Zinc-lead Sulphides.

Sampling operations being common ground, and by no means peculiar to the treatment of ores of lead, all reference to them has been omitted, but it has been thought best to retain a short summary of works' methods of assaying and analysis, which has been relegated to the "Supplementary" Section. In other respects the general arrangement has been retained, and so has the condensation of information into tabular form, which was made a prominent feature of the First Edition and has met with general approval. Brief descriptions of practice now obsolete have been retained in a few cases, either for completeness, or because the practice rendered obsolete in one locality might be worthy of adoption under different circumstances in another.

Published literature has been laid under contribution, so far as it has been available to the Author, particular use having been made of articles and summaries which have appeared in the various volumes of the *Mineral Industry*, in the *Engineering and Mining*

*Journal*, and the *Mining and Scientific Press*, also of papers published in the *Transactions of the Institution of Mining and Metallurgy*, and the *American and Australian Institutes of Mining Engineers*. The Author desires to express his gratitude to all those gentlemen in charge of metallurgical works who have kindly assisted him with information, particularly to Messrs. L. Guillaume, A. Eilers, W. H. Howard, A. K. Brewer, M. W. Iles, E. S. Weinberg, L. Pitblado, A. S. Dwight, A. E. Savage, P. Ninnis, and many others too numerous to mention by name.

No technical work can claim to treat exhaustively every branch of its subject, but it is hoped that the present volume covers the ground sufficiently well to merit from all interested in the Metallurgy of Lead a reception as cordial as that accorded to its predecessor; and that it will be found specially useful by the busy working metallurgist, to whom, as the Author knows from experience, in the absence of an extensive reference library the most copious Bibliography is of little value.

HENRY F. COLLINS.

HUELVA, *October, 1910.*

## P R E F A C E.

---

The object of Part I. of this treatise is to supply a condensed compendium of information on the Metallurgy of Lead, accurate as well as fairly up-to-date, and useful, therefore, to the busy working metallurgist as well as to the student. As regards American and German practice, the ground has been so well covered by the recent works of Hofman and Schnabel, that little has been attempted beyond condensing the information contained in the above works, supplementing it from other contemporary sources and from the Author's own practical experience and observation, and presenting a digest of the whole in a compact form.

In addition, however, Australian practice in smelting and in desilverisation will be found here described for the first time in considerable detail. It is further hoped that the standpoint aimed at in considering the whole subject, which is cosmopolitan rather than one mainly American or German, will render the information supplied more useful to English students and metallurgists.

With the object of economising space, as well as of facilitating reference and comparison, details of the practice at particular localities have been, as far as possible, thrown into the form of tabular statements. These will be found scattered throughout the book in almost every chapter, and it is hoped they will prove a useful feature.

Free use has been made of published literature on the subject, so far as it has been available to the Author. The source of all contributions is acknowledged in the footnotes, but special acknowledgment should be made to the works of Hofman (*Metallurgy of Lead*, 2nd edition, New York, 1893) and Schnabel (*Handbuch der Metallhüttenkunde*, Berlin, 1894), to the *Engineering and Mining*

*Journal* of New York, the *Transactions of the American Institution of Mining Engineers*, the *Columbia College School of Mines Quarterly*, and the *Berg- und Hüttenmännische Zeitung*; also to those friends who have supplied statistical information, and to many gentlemen in charge of metallurgical works, who, although personal strangers, have very kindly afforded the Author most valuable assistance.

Errors may have crept in, partly owing to the exigencies of a busy professional life, partly through the Author's absence from civilisation, which rendered it impossible for him to read all the proof-sheets. For all such shortcomings, he begs indulgent consideration at the hands of those who may have occasion to read or consult these pages.

HENRY F. COLLINS.

MINA DE STA. FE,  
CHIAPAS, MEXICO,  
*June*, 1899.

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# THE METALLURGY OF LEAD.

## SECTION I.

### CHAPTER I.

#### THE PROPERTIES OF LEAD AND ITS PRINCIPAL COMPOUNDS.

LEAD, as a material for the manufacture of sheet and pipe, has been known from very early times, and it was largely worked in Greece and in Spain many centuries before the Christian Era. The metal is occasionally found native in globules or thin films, but not crystallised; crystals artificially prepared by slow cooling are octahedra.

**Physical Properties.\***—*Colour, &c.*—Lead is bluish-grey in colour, and a freshly-cut surface shows a bright lustre, which rapidly tarnishes on exposure to the air. It is tasteless, but emits a slight odour when rubbed between the fingers.

**Hardness.**—Lead figures at only 570 on Bottone's scale, lying between tin and thallium, and far below all the other common metals; it is so soft as to be readily scratched by the finger nail, while it readily marks paper with a grey streak. Only the pure metal, however, exhibits this quality to perfection, the commercial article contaminated with Sb, As, Cu, and Zn being distinctly harder, while the degree of hardness increases with the amount of impurity present.

**Malleability and Ductility.**—Lead is very malleable, being readily rolled into thin foil; according to Spring,† lead filings can be formed into a solid block under a pressure of 13 tons to the square inch, while two and a-half times that pressure liquefies the metal. It is, however, low in the scale of ductility, owing to its want of tenacity, which prevents it from being drawn into fine wire, and which, indeed, is lower than that of any other common metal, being equal to only about 1 kilo. per sq. mm. of sectional area. On account of its softness and malleability lead may be readily squeezed out into solid or hollow cylinders by sufficient pressure applied at a temperature below its melting point, and this method is now usually adopted for the manufacture of both lead pipe and of solid rods.

\* v. Percy, *Metallurgy of Lead*, 1870, where the properties of this metal are given in great detail.

† Quoted by Roberts-Austen, *Introduction to the Study of Metallurgy*, 4th edition, 1898, p. 79.

**Specific Gravity.**—Lead is the heaviest of all the common metals, its specific gravity varying between 11.254 and 11.395, according as it is cast or rolled. Knab gives 11.352 and Reich 11.37 as the specific gravity of pure cast lead at 0° C., but the specific gravity of the commercial article is always lower on account of the impurities.

**Fusibility and Volatility.**—According to Le Chatelier the melting point of lead is 326° C.\* and according to Carnelly and Williams it boils between 1,450° and 1,600° C.,† but cannot be distilled. If, however, a mixture of zinc and lead be subjected to distillation at a high temperature, the zinc vapour carries over with it a considerable quantity of lead vapour; hence the source of part of the losses in lead smelting. When cooled slowly from the melting point, lead crystallises in groups of imperfect octahedra.

**Conductivity.**—Taking the conductivity of silver for heat as 100, that of lead at 12° C. is only 8.5, and with the same standard its conductivity for electricity is 10.7. Taking Hg at 0° C. = 1, the conductivity of lead is  $\frac{1}{818}$ .‡

**Specific Heat.**—The specific heat of lead between 10° and 100° C. was determined by Regnault to be 0.0314, while between - 77° and 10° it is 0.03065. The specific heat of molten lead between 350° C. and 450° C. was found by Porson to be 0.0402. Stücker§ finds it to be 0.03055 between 20° and 100° C., while above that temperature it gradually rises, being 0.03141 at 200° and 0.03289 at 300°.

**Other Physical Constants.**—The latent heat of fusion of lead is 5.369, its linear coefficient 0.0000292, and its coefficient of cubical expansion for 1° C. 0.000089.

The atomic weight of lead is 206.4.

**Chemical Properties.**—Lead undergoes no change in dry air or in water perfectly free from air, but is rapidly tarnished by air containing moisture and by water containing dissolved air; the dull grey coating which forms on its surface being probably the suboxide,  $Pb_2O$ , which afterwards becomes converted into carbonate. Melted at a low temperature in contact with air, iridescent pellicles of suboxide form on the surface, which at a dull red heat are converted into  $PbO$ ; and this  $PbO$ , when removed from the lead by continued exposure to the air at the same temperature, say 400° to 450° C., forms the red oxide or minium,  $Pb_3O_4$ . Other oxides of little importance are the sesquioxide,  $Pb_2O_3$ , and the peroxide,  $PbO_2$ .

Lead is readily dissolved by moderately dilute nitric acid, but is little affected by dilute  $HCl$  and by  $H_2SO_4$  of less than 60° B.; these acids form coatings of  $PbCl_2$  and  $PbSO_4$  respectively, on the surface of each particle, which protect it from further action. Even strong  $HCl$  in the cold and concentrated  $H_2SO_4$  at temperatures up to 200° have little effect on the metal. Boiling concentrated  $HCl$  and  $H_2SO_4$ , however, attack lead slowly even in large masses, and dissolve it with fair rapidity when in a fine precipitated condition. At 260° lead is dissolved completely by  $H_2SO_4$  with liberation of  $SO_2$  and free S; the action, however, is retarded by the addition of 1 per cent. Sb to the lead. Nitrous sulphuric acid acts upon it at all temperatures more readily than pure acid. Nitric acid itself, both strong and dilute, acts upon

\* *Eng. and Min. Journ.*, Oct. 11, 1890.

† *Roberts-Austen, op. cit.*, p. 73.

+ *Journ. Chem. Soc.*, xxxv., p. 563.

§ *Metallurgie*, 1906, iii., p. 487.

the metal readily at all temperatures and is its best solvent. Acetic, citric, tartaric, and other organic acids act upon it slowly in contact with moist air.

The compounds of lead are poisonous practically in proportion to their solubility, thus the metal itself and its sulphide, being incapable of absorption as such into the system, are practically innocuous, while the soluble salts, chloride, nitrate, acetate, &c., are active irritant poisons. The oxide, sulphate, and carbonate are much less active, but continued exposure to lead fumes or to any atmosphere in which these substances are present in the form of dust brings on at first "lead colic," and later, if exposure to the dust or fume is continued, a more chronic form of poisoning called "lead-palsy," accompanied by emaciation, deposition of lead sulphide in joints, gums, &c.

**Alloys of Lead and Impurities in the Commercial Article.\***—Two kinds of lead are known in the market—viz., *soft and hard*, or antimonial lead. The former is in part produced direct from pure non-argentiferous ores like those of the Mississippi valley, though chiefly from the refining of argentiferous lead; the latter is always a bye-product of the refining operations. Soft lead is employed for making sheet lead, pipe, shot, &c., for the preparation of pewter and solder, and for the manufacture of white lead. Hard lead is only used for making type metal, Babbitt metal, and other alloys, the demand for which purposes is variable, and at times insufficient to absorb even the comparatively small quantities produced.

The chief impurities in commercial lead are in the approximate order of their quantity Sb, Cu, Bi, Fe, Ag, Zn, Ni, Cd, As, Co, Mn, and S. The proportions in which they are found are given in the following table of analyses (Table I.). It is an interesting fact that all the impurities in a bar of commercial lead, except silver, are concentrated towards the top of the bar; this matter will be dealt with in greater detail in Chap. x.

*Silver.*—This metal alloys well with lead in all proportions, but, according to Levoll,† none of the alloys are quite homogeneous, a fact which, while it complicates the sampling of lead pigs containing silver,‡ goes to show that there is no definite chemical alloy between these metals, but that each dissolves the other in a molten condition. The affinity of lead for silver is so great that it removes the latter metal from its salts and combinations with sulphur, arsenic, and even from its alloy with copper. As a rule, commercial lead contains from  $1\frac{1}{2}$  to 12 dwts. of silver per ton, and rarely as much as 1 oz. The small quantities present appear to protect the lead somewhat from the action of sulphuric acid, but have no other perceptible effect.

Three alloys of silver and lead have been proposed for use in testing furnace temperatures; they are those containing the following numbers of parts by weight, namely, 9 Pb 1 Ag melting at 400° C., 3 Pb 1 Ag at 500° C., and 3 Pb 2 Ag at 600° C.

*Antimony.*—Only one definite alloy of lead and antimony is known—namely, the eutectic with 13 per cent. Sb, but the metals appear to mix in all proportions. Less than 0.005 per cent. of antimony has no apparent effect, larger amounts increase the hardness of lead, diminish its malleability, and render it unfit for white lead making.§ Lead with 0.1 per cent. Sb resists

\* v. Percy, *Metallurgy of Lead*, 1870; also Hofman, *Metallurgy of Lead*, N.Y., 1892.

† Percy, *Metallurgy of Silver*, p. 173.

‡ v. Chap. x.

§ Hampe, *Zeitschr. f. B.-H.-u. S.-wesen*, vol. xviii., p. 209; also Lunge, *E. and M. J.*, Jan., 1893.

TABLE I.—ANALYSES OF VARIOUS BRANDS OF COMMERCIAL LEAD.

Reference,	1	2	3	4	5	English Chemical.	B.H.P. Port Pirie, Australia.	B.H.P. Port Pirie, Australia.	R.H.P. Port Pirie, Australia.	Lone Elm, Missouri.	St. Joseph, Missouri.	Argentine, Kansas.	Selby, California.	Trail, B.C.
Process.	A	C	D	D	A	?	D	D	D	B	D	D	D	E
Ph,	99-98299	98-9841	99-9903	99-9980	99-9851	99-9693	99-9913	99-9853	99-97201	99-91680	99-99249	99-99249	99-9579	99-9890
Cu,	00069	0024	0015	0005	none	0097	0004	...	01782	06400	00022	00022	...	0003
Sb,	tr.	0056	0030	0030	0160	0149	0051	0108	00146	00130	00127	00127	...	0020
As,	tr.	...	0005	...	...	0002	tr.	...	00011	00080	tr.	tr.	0024	...
Bi,	...	0023	...	0007	0048	tr.	...	...	00056	00097	00308	00308	0300	...
Ag,	00025	0018	0004	0010	0015	0020	0008	0009	00056	00020	00020	00020	0010	0025
Fe,	00055	0017	0008	0047	0026	0029	0015	0025	00686	00100	00178	00178	0027	0053
Mn,	...	...	...	...	...	...	...	...	...	...	...	...	0008	...
Ni,	...	0011	0005	...	tr.	0003	...	...	00077	01600	tr.	tr.	0001	tr.
Co,	...	...	...	...	tr.	tr.	tr.	tr.	00005	...	...	00075	...	0002
Zn,	00076	0010	0030	0021	tr.	tr.	0009	0001	00033	...	...	...	tr.	0007
Cd,	...	...	...	...	...	0007	tr.	tr.	...	...	...	...	0004	...
Sn,	...	...	...	...	...	...	tr.	0004	...	...	...	...	...	0007
S,	01476	...	...	...	...	...	tr.	tr.	...	...	...	...	...	tr.
Insol.,	...	...	...	...	tr.	tr.	...	tr.	...	...	...	...	0040	tr.
	100-00000	100-0000	100-0000	100-0000	100-0000	100-0000	100-0000	100-0000	100-00000	99-99997	100-00000	100-00000	99-9996	100-0000

Note.—In all the above analyses, except Nos. 10 and 12, lead has been determined by difference.

Process.—A = Reverberatory; B = Hearth; C = Blast-furnace followed by Pattinson; D = Blast-furnace followed by Parkes' process; E = Electrolytic.

References.—1. *Oesterr. Jahrbuch des K. K. Berg-Akademien*, vol. xxii., p. 389. 2. *Oesterr. Zeitschrift f. B. u. H.-wesen*, 1885, p. 208. 3. *Firket, Annales des Mines de Belgique*, 1901, p. 237. 4. *Ferraris, Oesterr. Zeitschr. f. B. u. H.-wesen*, p. 455, and *E. and M. J.*, Oct. 28, 1905, p. 784. 5, 6, 8, 12, and 13. *Engineering and Mining Journal*, vol. lxxxiii., June 22, 1907, p. 1179. 7. *Private Notes*, 1896. 9. *Trans. A.I.M.E.*, vol. xviii., p. 687. 10. *Mtn. Ind.*, xiii., 1905, p. 274. 11. *Engineering and Mining Journal*, July 14, 1882.

the action of cold sulphuric acid better than pure lead, but is more easily attacked by hot acid (*Lunge*). With 0·2 per cent. Sb this difference becomes more pronounced, and hot acid has then a marked effect upon the metal. A bar of lead containing a small proportion of antimony shows in the centre a spot with uneven surface resembling moss. On heating to a bright red with access of air, antimony (and arsenic) are slagged off as antimoniate,  $\text{Pb}_3(\text{SbO}_4)_2$  (and arseniate), of lead. This reaction comes into play in the softening and refining of lead for market. For the manufacture of white lead the metal should be as free as possible from antimony. As little as 0·05 per cent. Sb has a marked effect in retarding the process without affecting the colour of the product, while 0·1 per cent affects the colour also.

Alloys of antimony and lead, being much harder than pure lead, are in demand for pumps, nozzles, and other portions of apparatus for resisting the action of acid solutions. Up to a content of 10 per cent. Sb, these alloys can be rolled satisfactorily, provided the casting into cakes has been conducted at the lowest possible temperature, only just above the melting point of the alloy.

Pure lead melting at  $326^\circ$ , the eutectic alloy with 13 per cent. Sb at  $228^\circ$ , and pure antimony at  $632^\circ \text{C.}$ , the determination of the melting point serves as a fairly accurate method of determining the percentage of antimony in hard lead, provided that no other metals be present, and that it is known before-hand that the alloy is either richer or poorer than the eutectic.

*Arsenic.*—Lead and arsenic appear to mix freely in all proportions up to 34 per cent. As, but form no definite chemical compounds. The eutectic alloy contains 2·5 to 3·0 per cent. As, and freezes at  $292^\circ \text{C.}$

*Bismuth.*—Anything over 0·075 per cent. of bismuth affects the quality of lead for white-lead making, although, according to Endemann,\* the presence of bismuth favours corrosion, and the colour of the product is not affected, provided the process be stopped before completion, any bismuth present being then concentrated in the small blackish residue. According to Napier,† 0·1 per cent. of bismuth acts like copper and antimony in protecting lead against the action of cold sulphuric acid. In larger quantity, 0·1 to 0·35 per cent., bismuth makes lead slightly harder, somewhat crystalline, and more fusible; though, according to the researches of Burggraf,‡ its malleability is not affected by admixture of bismuth, even in proportions up to 2 per cent. When present, bismuth is not removed from lead by the Parkes process, but may be to a great extent separated by slow cooling of the molten lead, which produces crystals purer than the surrounding medium, as in the Pattinson process (which see).

*Copper.*—Copper forms no definite alloys with lead, for, although the two metals may be melted together in all proportions, part of the lead (containing, however, upwards of 2 per cent. copper) will, according to Percy's experiments, always separate out on slow cooling, leaving the copper (retaining lead up to as much as 25 per cent.) in the condition of a porous mass. Lead containing 1 or 2 per cent. of copper can be freed from that metal by carefully melting down at a low temperature and skimming; the residual lead, however, always retains about 0·08 per cent. § Copper may be more perfectly separated

\* *American Chemist*, 1876, vol. vi., p. 257.

† *Chem. News*, 1880, p. 314.

‡ Quoted by Schnabel, *Handbuch der Metallhüttenkunde*, vol. i., p. 277.

§ Hofman, *op. cit.*, p. 22; also Huntington, *E. and M. J.*, March 30, 1895.

by means of zinc, as in Parkes' process (which see). Lead for making white lead or flint glass should not contain more than '0014 of copper,\* but a larger proportion appears to have but little effect when it is used for other purposes; like antimony, a small quantity of copper protects lead from the action of cold  $\text{H}_2\text{SO}_4$ , but by hot and concentrated acid lead with from 0.1 to 0.2 per cent. copper is as readily attacked as is the pure metal.

*Iron.*—The small quantity of iron usually found in commercial lead, up to a maximum of, say, 0.05 per cent., exercises no influence upon its physical properties. Above 0.003 per cent. would, according to Landsberg,† unfit the lead for white-lead making. Lead and iron form no alloys, having no affinity for each other.

*Zinc.*—This metal, even in small quantities, gives lead a white colour and makes it so hard as to be unfit for rolling, rendering it, moreover, easily attacked by both cold and hot  $\text{H}_2\text{SO}_4$ . The limit of zinc for white-lead making is the same as that of iron. Lead and zinc form no definite alloys, and cannot, like lead and copper, be melted together in all proportions. Rössler and Edelmann‡ have determined the solubility of zinc in molten lead at various temperatures, and find it to vary from 0.6 to 0.8 at 400° C., up to 3 per cent. at 700° C. Heyn§ found that, when zinc and lead were stirred together and allowed to rest, they invariably separated into two strata, which more nearly represented the pure metals the lower the temperature, but that even at 419° C. the upper stratum of zinc contained 1.5 per cent. Pb, while after the upper stratum had set the lower stratum at 334° still contained 1.2 per cent. of zinc. In the refining of zinc obtained by distillation in Silesian furnaces, while a lead alloy containing 4.24 per cent. Zn is separated out in the bottoms of the kettles, the supernatant zinc still carries 1.12 per cent. lead.

*Tin.*—This metal is of very uncommon occurrence in commercial lead, but when present it hardens the lead, increasing its fusibility and its liability to attack by acid. Like antimony and arsenic, it is readily removed by oxidation at a bright red heat, and it is, in fact, the most readily oxidisable of the three metals. Many supposed alloys of lead and tin have been described, but, according to Wiesengrund,|| only that with 63 per cent. Sn, corresponding to the formula  $\text{PbSn}_3$ , can be considered a definite chemical alloy. It solidifies at 178½° C., and melts at 183° C., the melting point being raised by addition of either of the component metals, and its specific gravity is less than that indicated by its percentage composition.

*Aluminium.*—Aluminium and lead form no definite alloys, and although melted and stirred together, they separate out almost completely on cooling, the lead retaining less than 0.5 per cent. Al, while the solidified aluminium on top of the still molten lead contains a little more of the latter metal. The aluminium retained by the lead is very readily removed by oxidation (see the Rössler-Edelmann process of desilverisation).

*Calcium.*—According to Harkspill,¶ alloys of lead and calcium can be made by introducing the latter metal into fused  $\text{PbCl}_2$ , or by electrolysing molten  $\text{CaCl}_2$  with a lead cathode. A definite compound is stated to be

\* Hampe, *loc. cit.*

† Quoted by Schnabel, *op. cit.*, 24.

‡ *E. and M. J.*, Nov. 15, 1890.

§ *B. u. Hütt. Zeit.*, lix., 47, 559, Nov. 23, 1900.

|| *Berg.-u. Hüttenmännische Zeitung*, 1894, p. 305.

¶ *Comptes Rendus*, 1906, cxliii., 227.

formed, having the formula  $Pb_3Ca_{11}$ , with melting point  $775^{\circ} C.$ , and a specific gravity of 7.6.

**Sodium.**—Mathewson,\* who investigated the alloys of lead and sodium, found four chemical compounds—viz.,  $Na_4Pb$ ,  $Na_3Pb$ ,  $Na_2Pb_2$ , and  $Na_2Pb_3$ —freezing respectively at  $386^{\circ}$ ,  $405^{\circ}$ ,  $367^{\circ}$ , and  $319^{\circ}$ , and four eutectics—viz.,  $Na_4Pb + Na_3Pb$ ,  $Na_3Pb + Na_2Pb_2$ ,  $Na_2Pb_2 + Na_2Pb_3$ , and  $Na_2Pb_3 + Pb$ —freezing respectively at  $373^{\circ}$ ,  $329^{\circ}$ ,  $301^{\circ}$ , and  $307^{\circ}$ .

**Tellurium.**—Lead appears to alloy readily with tellurium in almost all proportions. There appears to be only one eutectic, which freezes at  $400^{\circ} C.$ , it contains  $21\frac{1}{2}$  per cent. Pb, equivalent to a formula of  $PbTe_6$ .†

**Selenium.**—According to H. Pélabon,‡ a definite compound exists with a formula  $PbSe$  (27.65 per cent. Se), which freezes at  $1,065^{\circ} C.$

**Antifriction Alloys.**—The best antifriction alloys are composed chiefly of tin with certain proportions of antimony and copper, but they are too expensive for many purposes, and cheaper alloys for similar work have lead as their principal basis, with admixtures of antimony and tin. A new antifriction alloy, composed of Pb 98.75 per cent., Mg 1.25 per cent., is prepared by melting lead under a cover of fluorspar and stirring in magnesium; the resulting alloy may be remelted in ladles and poured in the usual way. Another, the so-called “tempered lead,” is simply lead with 2 per cent. sodium added when heated slightly above its melting point under a cover of oil. This alloy must be always melted under a cover of oil or resin, its great advantage is the formation of soap by oxidation of the sodium and saponification of the lubricant, by which means the bearing is always kept cool with only a small consumption of oil.

**Alloys of Industrial Importance.**—Table II. gives the approximate composition of a number of the principal commercial alloys of lead; it will be seen that there are wide variations in composition amongst alloys which pass under the same name.

A good method for the determination of lead in alloys will be found in Chap. xix.

#### COMPOUNDS OF LEAD.

**Lead Oxide,  $PbO$ .**—This, the simplest oxide, is also the only one of any metallurgical importance. It occurs in two distinct forms as *massicot* and *litharge*. The former is an amorphous yellow powder produced by exposing melted lead at a dull red heat to the action of air, continually removing the film of suboxide, and oxidising it to the yellow oxide at a low temperature, avoiding fusion. On raising the temperature to a bright red heat the oxide melts, and, on cooling, solidifies as litharge, which varies in colour according to the rate of cooling, slow cooling bringing out a red, and rapid cooling a yellow colour. On a commercial scale, litharge is formed by cupelling argentiferous lead. The molten litharge produced is of an orange colour and transparent while hot; when run out from the furnace over an iron plate, or when dropped into water, it forms opaque, greasy-looking, yellow lumps. When allowed to form large cakes of a ton or more in weight, and to cool slowly, the outer solid crust is of the usual yellow colour, while the interior

\* *Mineral Industry*, vol. xv., 1907, p. 523.

† Fay & Gillson, *Amer. Chem. Journ.*, 1902, No. 27, p. 81.

‡ *Comptes Rendus*, 1907, cxliv., 1159.



TABLE II.—ALLOYS OF LEAD.

	Pb	Sb	Sn	Cu	Bi	Zn	As	Na	Fe
Type metal, . . . . .	55	30	15	...	...	...	...	...	...
" . . . . .	60	25	15	...	...	...	...	...	...
" . . . . .	70	18	10	2.0	...	...	...	...	...
" . . . . .	55	23	22	...	...	...	...	...	...
" . . . . .	72	18	25	...	...	...	...	...	...
Stereotype metal, . . . . .	82	15	.3	...	...	...	...	...	...
" . . . . .	86	14	...	...	...	...	...	...	...
" . . . . .	70	15	...	...	15	...	...	...	...
Linotype metal, . . . . .	84.5	13.5	2	...	...	...	...	...	...
Bearing metal, heavy loads, .	84	16	...	...	...	...	...	...	...
" . . . . .	88	12	...	...	...	...	...	...	...
" . . . . . very heavy, .	42	...	15	3	...	40	...	...	...
" . . . . . " . . . . .	4	6	38	1	...	47	...	...	...
" . . . . . light loads, .	60	20	20	...	...	...	...	...	...
" . . . . . " . . . . .	42	16	42	...	...	...	...	...	...
" . . . . . American Rys., .	88.9	7.4	...	3.7	...	...	...	...	...
" . . . . . " . . . . .	78.4	20	...	...	...	1	...	...	0.6
" . . . . . Magnolia, .	78.3	17.8	3.9	tr.	...	...	...	...	...
" . . . . . " . . . . .	80	15	5	...	...	...	...	...	...
" . . . . . " . . . . .	83.5	16.5	...	...	tr.	tr.	...	...	...
Shot metal, . . . . .	99.6	...	...	...	...	...	0.2 to	...	...
Pewter, . . . . .	20	...	80	...	...	...	0.35	...	...
" . . . . . Queen's metal, .	9	9	73	...	9	...	...	...	...
Britannia metal, . . . . .	2	7	89	2	...	...	...	...	...
" . . . . . " . . . . .	1	10	80	9	...	...	...	...	...
" . . . . . " . . . . .	...	6	90	2	...	...	...	...	...
Solder, plumbers', . . . . .	66.6	...	33.3	...	...	...	...	...	...
" . . . . . common, .	50	...	50	...	...	...	...	...	...
" . . . . . fine, . . . . .	33.3	...	66.6	...	...	...	...	...	...
Fusible metal, Newton's, .	31	...	19	...	50	...	...	...	...
"No-heat" metal, . . . . .	98	...	...	...	...	...	...	2	...
Acid-resisting hard lead, .	78	22	...	...	...	...	...	...	...
White metal for rope-capping,	60	9	30	...	1	...	...	...	...

swells and breaks up into loose red flakes. The swelling is caused by expulsion of oxygen which has been absorbed by the molten litharge and is given off on cooling. The specific gravity of litharge is 9.36, its melting point according to Honsell\* is 954° C., or slightly above the melting point of silver; in its molten condition it is a good conductor of electricity. According to Doeltz and Graumann† litharge becomes liquid at 900° C., it begins to volatilise slightly at 800° C., and the rate of volatilisation increases rapidly at temperatures over 1,000° C. According to Mostowitsch‡ the melting point of PbO is only 883° C. Litharge is very slightly soluble in water (1 part in 12,000), but dissolves readily in nitric and acetic acids.

Litharge is a strong base, rapidly corroding bricks and clays, and forming the silicates described below; it also forms readily fusible compounds with all the metallic oxides. With potash and soda and the alkaline earthy bases it apparently acts as an acid, with stannic oxide and arsenic and antimonie acids, as a base, while in other cases it seems to act simply as a solvent. The

\* *Berg- u. Huttenmännische Zeitung*, 1866, p. 106.

† *Metallurgie*, 1906, iii., p. 406.

‡ *Metallurgie*, 1907, iv., p. 648.

following table\* shows the proportions of litharge required to form fusible compounds with the principal metallic oxides :—

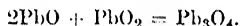
1 part of	Cu <sub>2</sub> O.	CuO.	ZnO.	Fe <sub>3</sub> O <sub>4</sub> .	Fe <sub>2</sub> O <sub>3</sub> .	MnO.	SnO <sub>2</sub> .	Sb <sub>2</sub> O <sub>3</sub> .	Sb <sub>2</sub> O <sub>4</sub> .	As <sub>2</sub> O <sub>3</sub> .	As <sub>2</sub> O <sub>5</sub> .
Requires parts of litharge, }	under 1·5	under 1·8	8	4	10	10	12	any	5	4 to 8	25 to 1·0

Litharge is easily reduced to the metallic state by all reducing agents (by CO at only 100° C., by H at 310° C.),† and, therefore, forms a valuable oxidising agent. Reduction by carbon commences at from 530° to 555°, becoming very decided at 600° C.‡ When mixed with S, Te, As, Sb, Bi, Cu, Zn, or Fe, these substances become wholly or partially oxidised, and their oxides either volatilise or dissolve in the excess of litharge, a corresponding amount of lead being reduced and combining with any unoxidised portion of the added element.

**Lead Suboxide, Pb<sub>2</sub>O.**—When lead is heated at a low temperature a grey pellicle forms upon its surface, which is supposed to be chiefly the suboxide. A comparatively pure suboxide is prepared as an intermediate stage in the Bischof white-lead process (*q.v.*) by reduction of PbO by means of water-gas. As thus prepared, the suboxide is a velvety-black powder in an exceedingly fine state of subdivision, the composition of which, however, is by no means constant.

**Minium, or Red Lead, Pb<sub>3</sub>O<sub>4</sub>.** may be regarded as a combination of 2 mols. PbO with 1 mol. PbO<sub>2</sub>. It is a bright-red powder formed by submitting the amorphous simple oxide (massicot) to oxidation at a temperature of about 300° C. At a higher temperature it loses oxygen, becoming reconverted into PbO. According to Matuschek§ pure Pb<sub>3</sub>O<sub>4</sub> free from litharge may be prepared in small quantities by adding KClO<sub>3</sub> to fused KNO<sub>3</sub>, and deflagrating lead into the mixture, which is then heated until no more gas is evolved, and boiled out with water.

Another method of preparing minium is to heat together a mixture of lead peroxide and litharge, when they combine according to the equation—



According to Dorsemagen|| the best temperature for the reaction is 250° C., and it is best to employ a slight excess above the theoretical proportion of litharge, when, after grinding the product wet, it is found to have a fine colour.

**Lead Peroxide, PbO<sub>2</sub>.** may be prepared by digesting minium with dilute nitric acid until no more is dissolved, or by acting upon a solution of lead acetate with calcium hypochlorite. The amorphous powder left in an insoluble condition has the above composition when washed and dried; it

\* Hofman, *op. cit.*, p. 16, quoted from Percy, *op. cit.*, pp. 17 to 18.

† Roscoe and Schorlemmer, *Treatise on Chemistry*, 1892, vol. ii., part i., p. 282. According to Mostowitsch, reduction commences between 290° and 300° (*Metallurgie*, 1907, iv., p. 648).

‡ Doeltz and Graumann, *Metallurgie*, 1907, iv., p. 420.

§ *Chem. Zeitung*, 1905, xxxix., p. 510.

|| *Zeitschr. f. Elektrochemie*, VIII., xxxix., 745, Sept. 25, 1902.

acts as a feeble base with acetic acid, but as an acid with oxides of the alkali and alkaline earthy metals, forming compounds which are called plumbates. It is a powerful oxidising agent, and, as it detonates with phosphorus, it is utilised in the manufacture of lucifer matches. It is used in the construction of storage batteries, and is alternately destroyed and re-formed during the operations of discharging and re-charging them.

Lead peroxide may be economically prepared by the following process\* :—Lead sulphate mixed with magnesia is suspended with water in a revolving barrel, and chlorine is passed in to saturation during about three hours. The proportions are for each 5 kilos. of lead sulphate, magnesia 1.2 kilos., water 13 litres, and chlorine 1.2 kilos. The crude reaction product contains already about 90 per cent. of peroxide, and by washing first with caustic soda and then with acid a product is obtained containing only traces of Cl and 97 per cent. of  $PbO_2$ . The cost of production is said to be only 44 francs per 100 kilos. (= 1s. 9d. per lb.).

**Lead Carbonate,  $PbCO_3$ .**—Lead carbonate occurs in nature as the mineral cerussite (sp. gr. 6.46). At  $200^\circ$  it is decomposed into  $PbO$  and  $CO_2$ , and at the same temperature is reduced to metallic lead by  $CO$ , charcoal only acting upon it at a somewhat higher temperature.

**White Lead**, the hydrated basic carbonate,  $2PbCO_3 + PbH_2O_3$ , an important article of commerce, is ordinarily prepared by the "Dutch," or "corroding" process, which consists in acting upon lead plates or gratings by the vapour of acetic acid and  $CO_2$  generated from decomposing organic matter, and removing the white crust formed from the residual lead; full descriptions of the process will be found in works on Chemical Technology. Attempts have been made to shorten the above process, which lasts many weeks.

The process of the *M<sup>r</sup> Dougall White Lead Co.*, of Buffalo,† comprises the following operations :—

1. Melting lead and granulating it in a brick chamber by jets of steam, which cut the streams of molten lead at an angle of  $45^\circ$ . The product is put through a fine screen, and the over-size returned.

2. From 15 cwts. to a ton of the granulations are put into a drum with 176 lbs. of 40 per cent. acetic acid, which is added in three equal instalments—viz., on the 1st, 3rd, and 5th days.

3. The drum is rotated for seven days, steam, air, and filtered fuel gases being blown through at intervals, the moisture being only sufficient to form a paste.

4. The product is run through a pug-mill, then down riffled launders to catch any grains of unaltered lead, then treated with caustic soda in settling tanks, and dried.

The *Bischof* process worked at Brimsdown, near London, comprises the following operations :—

1. Oxidation of metallic lead to litharge in gas-fired English cupellation furnaces.

2. Crushing of the litharge and reduction to  $Pb_2O$  at a temperature of

\* *Moniteur Scientifique*, 1906, xx., pp. 514-518 (*E. and M. J.*, Sept. 8, 1906, p. 437).

† *E. and M. J.*, Oct. 21, 1905, p. 745.

300° C. in closed retorts by means of water-gas generated in vertical retorts by blowing steam through red-hot charcoal.

3. Hydration of the  $\text{Pb}_2\text{O}$  in open-topped pans with agitators, yielding a smooth paste of  $\text{PbH}_2\text{O}_2$ .

4. Carbonation of the lead hydrate, in closed wooden vats furnished with stirrers, by means of  $\text{CO}_2$  generated by boiling a solution of alkaline bicarbonate produced by absorption of fuel gases in a solution of sodium carbonate. Each vat contains a measured quantity of lead acetate solution, to which is added a weighed quantity of the hydrate paste, and gas is passed until samples show that all trace of yellow oxide has disappeared, and the percentage of carbonate in the product attains a constant.

5. The product is run through filter presses, washed with soda solution and clean water, and, without drying, charged into pug-mills, where it is combined with linseed oil, the resulting paste being ground in paint mills with granite rollers to eliminate air bubbles.

Unlike all other white lead produced by wet processes, the Bischof white lead has a greater covering power than the ordinary article in the proportion of 14 to 10.

**Lead Sulphide,  $\text{PbS}$ .**—This substance is found native as galena (sp. gr. 7.3 to 7.7), and it may be formed artificially by melting together lead and sulphur in proper proportions, by reducing lead sulphate with any reducing agent, and by precipitating any solution of a lead salt with  $\text{H}_2\text{S}$ .

Lead sulphide, although not so fusible as the metal, readily melts at a bright red heat (935° C. according to Lodin), becoming very fluid, and penetrating all kinds of furnace linings to a surprising degree. According to the same author,\* its vapour tension is considerable at temperatures much below this, which explains why it volatilises readily and in an unaltered condition when out of contact with air in lead furnaces, part of the vapour being carried away by the furnace gases, and part condensing to form the crystals and crusts of sublimed galena, which are of such common occurrence in the upper parts of lead furnaces. According to the experiments of Doeltz and Graumann,† lead sulphide volatilises rapidly at 860° C., at which temperature an experimental quantity lost in one hour about 18 per cent. of its weight in a current of nitrogen. According to Guichant,‡ it melts at 1,015° C., and boils at 1,085° C. According to Friedrich,§ the melting point of  $\text{PbS}$  is as high as 1,114° C., and this figure is probably correct, being accepted amongst others by Fulton and Goodner.||

Being isomorphous with  $\text{Ag}_2\text{S}$ ,  $\text{Cu}_2\text{S}$ ,  $\text{ZnS}$ ,  $\text{FeS}$ , &c., lead sulphide readily mixes with these sulphides in all proportions; such mixtures form the “mattes” obtained in smelting complex sulphides, which are described in detail in Chap. x. Both galena itself and lead matte are good conductors of electricity.¶ With the sulphides of antimony and arsenic lead sulphide forms sulpho-salts, both simple and complex, many of which occur in nature as well-marked mineral species—e.g., Zinkenite, Jordanite, Jamesonite, Bournonite, &c.

\* *Comptes Rendus*, 1895, vol. cxx., pp. 1164 to 1167, abstract in *Journ. Soc. Chem. Ind.*, vol. xiv., p. 807.

† *Metallurgie*, 1906, III., xiii., p. 406.

‡ *Comptes Rendus*, cxxiv., pp. 1224-1226.

§ *Metallurgie*, v., 1908, p. 51.

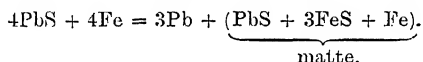
|| *Bull. Amer. Inst. M.E.*, Nov. 1908, p. 968.

¶ Kiliani, *Berg- u. Hüttenm. Zeitung*, 1883, pp. 237, 366, 378.

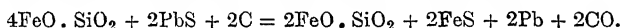
Many Continental metallurgists have advocated the existence of sub-sulphides of lead, but Percy's investigations showed that all such supposed subsulphides on careful heating eliquate metallic lead, and leave a residue of crystalline sulphide; further, that lead and its sulphide can be melted together in all proportions, the properties of the mixtures being intermediate between those of their components, and approximating to one or other according to the predominating constituent. Friedrich and Leroux\* have recently investigated the subject by means of microphotographs, and confirm Percy's conclusion that no intermediate compounds exist between PbS and Pb.

*Decomposition of Lead Sulphide.*—Galena is decomposed by fusion with iron, the reaction forming the basis of what is called the "precipitation" process of lead smelting; which, in a modified form, is still in use at Clausthal, and which, to a greater or less extent, comes into play in the furnace work of all modern blast-furnace lead-smelting plants. The reaction is often given as  $\text{PbS} + \text{Fe} = \text{FeS} + \text{Pb}$ , but, although more perfect the higher the temperature employed, the decomposition is never complete.

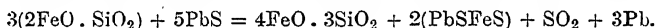
The maximum amount of lead obtainable from pure galena (containing 86.6 per cent. Pb) varies from 72 to 79 per cent., the rest being retained in the iron sulphide. Nolte† explained the reaction by supporting the formation of a subsulphide of iron, but the observed facts are easily explained on the far more probable supposition first published by Münster‡ that a portion of the iron remains dissolved in the residual iron sulphide, which also may dissolve a portion of the lead. The reaction may then be—



In order to make use of this reaction in furnace work it is necessary to employ a high temperature; but the metallic iron required may be reduced in the furnace itself from oxides or from the basic silicate,



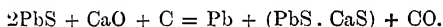
At a very high temperature the monosilicate (as also the more basic silicates) of iron reacts upon galena without the intervention of carbon, forming a sesquisilicate, according to the following equation given by Percy:—§



Both lime and baryta partially decompose galena; if the atmosphere be not reducing the reaction is probably



and in presence of carbonaceous matter, according to Berthier,||



Galena is only partially decomposed by fusion with copper, the reduced lead alloying with the unaltered part of the copper, while the  $\text{Cu}_2\text{S}$  formed mixes with the unaltered galena. Zinc and tin\* also effect only a partial

\* *Metallurgie*, 1905, ii., p. 536.

† *Berg- u. Hüttenm. Zeitung*, 1860, p. 165.

‡ *Ibid.*, 1877, p. 195.

§ *Op. cit.*, p. 59.

|| *Traité des essais par la voie sèche*, Liège, 1887, vol. ii., p. 580.

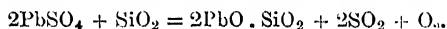
decomposition, the infusibility of the ZnS in the one case and the approximately equal affinities of lead and tin for sulphur in the other case preventing a proper separation of metallic lead.

Wiedmann,\* by tracing the freezing-point curves of mixtures of PbS and FeS, finds that they form an eutectic freezing at 782° C., with a mean composition FeS 26 per cent., PbS 74 per cent., some eutectic being always present in all mixtures varying between the extreme limits of 5 per cent. and 90 per cent. PbS.

Friedrich,† however, finds the freezing point of the eutectic of PbS and FeS to be 863° C., and its composition to be FeS 30 per cent., PbS 70 per cent. The same author finds that PbS forms with other sulphides the following eutectics:—

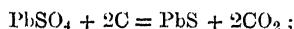
PbS 49 per cent. and Cu<sub>2</sub>S 51 per cent., freezing point 540° (Cu<sub>2</sub>S alone 1135° C.).  
 PbS 27 per cent. and Ag<sub>2</sub>S 73 per cent., freezing point 630° (Ag<sub>2</sub>S alone 835° C.).

**Lead Sulphate, PbSO<sub>4</sub>.**—This occurs in nature as the mineral anglesite (sp. gr. 6.2), and may be formed by a careful partial roasting of galena. It is the most stable of all the sulphates of the heavy metals, remaining unaltered at a bright red heat. According to the experiments of Doeltz and Graumann‡ lead sulphate is only very slightly volatile at 900° C., but it commences to dissociate at 1,000° C. At a white heat it softens and loses a part of its SO<sub>2</sub>, becoming basic sulphate. Although undecomposed by simple heating, it is readily decomposed by silica according to the equation—



Upon this reaction is based the process of “slag roasting,” hereafter to be described.

Carbon in proper proportions at a dark red heat (also carbonic oxide at a high temperature) completely reduces the sulphate to sulphide, according to the equation—



but if not in sufficient quantity, only the exact molecular proportion of sulphate corresponding to the weight of carbon present is reduced, the rest remaining unaltered. At a cherry red heat this unaltered sulphate reacts upon the reduced sulphide, as will be subsequently explained.

Lead sulphate is reduced at a high temperature by lead ( $\text{PbSO}_4 + \text{Pb} = 2\text{PbO} + \text{SO}_2$ ) and by iron ( $\text{PbSO}_4 + 4\text{Fe} = \text{Fe}_3\text{O}_4 + \text{FeS} + \text{Pb}$ ). It suffers a double decomposition with lime, yielding calcium sulphate. Lead sulphate is slightly soluble in water (1 part in 22,800), but considerably less so in dilute H<sub>2</sub>SO<sub>4</sub> (1 part in 36,500). It is, however, more soluble in HNO<sub>3</sub> and nitrate solutions, and in hot HCl, while concentrated H<sub>2</sub>SO<sub>4</sub> dissolves it to the extent of 6 per cent. It is also soluble in solutions of the alkaline chlorides, CaCl<sub>2</sub>, and calcium acetate; in all ammoniacal salts, especially the acetate; and slightly in sodium hyposulphite, the solubility varying with the temperature and concentration of each solvent.

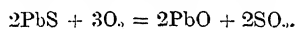
**Roasting of Lead Sulphide.—Reactions between Sulphide, Sulphate, and Oxide.**—When lead sulphide is roasted at a low temperature

\* *Metallurgie*, 1906, iii., p. 660-671.

† *Ibid.*, 1907, iv., pp. 479 and 671, and 1908, v., pp. 50, 52, 122.

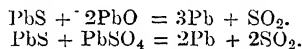
‡ *Ibid.*, 1906, III., xiii., p. 442.

it is at first simply oxidised with evolution of  $\text{SO}_2$  according to the equation—

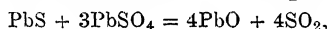


A part of the  $\text{SO}_2$  takes more oxygen from the air becoming  $\text{SO}_3$ , which combines with the  $\text{PbO}$  to form  $\text{PbSO}_4$ . According to Plattner,\* pure galena yields on roasting a much smaller proportion of sulphate to oxide than is given by galenas containing much pyrites; this is probably on account of the greater tendency of  $\text{SO}_2$  to form  $\text{SO}_3$  in presence of iron oxides. It is possible that some  $\text{PbSO}_4$  may be formed from  $\text{PbS}$  by direct absorption of oxygen, as suggested by Rammelsberg,† but this is doubtful.

When mixtures of lead sulphide with oxide or sulphate in suitable proportions are subjected to a bright red heat, at which the mixtures soften, the following reactions (amongst others) take place :—

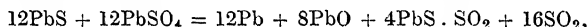


The limiting temperature at which the first of the above reactions begins to take place is, according to Lodin,‡  $720^\circ \text{C.}$ , and the second commences at  $670^\circ \text{C.}$  According to Schenk and Rassbach,§ the limiting temperatures for these reactions are respectively  $650^\circ$  to  $660^\circ$  for the first and  $550^\circ$  for the second, but both are retarded if the  $\text{SO}_2$  is not withdrawn as soon as made, and may even be reversed at a higher temperature under considerable pressure of  $\text{SO}_2$ . In the case of oxide the decomposition invariably takes place in molecular proportions, so that any excess of sulphide on the one hand or of oxide on the other remains unaltered on the surface of the lead, except in so far as it may have been affected by the material of the crucible, or other vessel in which the experiment has been made. When  $\text{PbS}$  is fused with excess of  $\text{PbSO}_4$ , however, another reaction takes place, viz. :—



and this reaction may take place simultaneously with the two preceding in different parts of the same charge. It is probable, indeed, that towards the end of the operation this reaction becomes developed very largely at the expense of the preceding. The limiting temperature for this reaction appears to be about  $550^\circ \text{C.}$  One very important process of lead extraction depends exclusively upon the above reactions, and, as will be seen hereafter, it is essential that the temperature be kept low during the roasting, and then raised suddenly in order to bring about the reactions.

According to Hannay,|| all the above reactions given by Percy and other authorities are incorrect. This author found that  $\text{PbS}$  is much more volatile at a white heat in  $\text{SO}_2$  than in other gases, and supposed the existence of a body to which he assigns the formula  $\text{PbS} \cdot \text{SO}_2$ , from which the  $\text{PbS}$  crystallises out on cooling. He supposed the principal reaction which takes place on heating together  $\text{PbS}$  and  $\text{PbSO}_4$  to be the following :—



and proposed to treat galena ores by melting and blowing in a kind of Besse-

\* *Die Metallurgischen Röstprozesse*, Freiberg, 1856, p. 145.

† Rammelsberg, *Die Metallurgie des Bleies*, 1872, p. 39.

‡ *Comptes Rendus*, cxx., 1895, pp. 1164-1167.

§ *Metallurgie*, 1907, iv., p. 455.

|| *Trans. Inst. Min. Met.*, vol. ii., p. 188.

mer converter to obtain (1) a small quantity of lead rich in silver; (2) molten litharge; (3) fume which by excess of air is converted into lead sulphate. Huntington\* regards the volatilisation of  $\text{PbS}$  in  $\text{N}$ ,  $\text{CO}_2$ ,  $\text{SO}_2$ , and other gases as probably a case of mere solution in a gas; and points out the impracticability of Hannay's proposals, especially as regards the condensation of the enormous amount of lead fume formed, and the difficulty of finding any furnace lining to withstand the action of molten litharge at the temperature of molten  $\text{PbS}$ . Lodin† also confirms the reactions heretofore universally recognised as being correct at the temperatures prevailing in ordinary lead furnaces. There is no doubt that Hannay's work is interesting, as showing that the volatility of  $\text{PbS}$  is greater than it is generally supposed to be, and that it is increased in the presence of certain gases; but his explanation of the reactions taking place in lead furnaces must be regarded as untenable, and his proposed "new metallurgy of lead" impracticable. Jenkins and Smith,‡ while entirely confirming the old reactions, prove that lead is attacked by sulphur dioxide. In the introduction to Section II., dealing with Lead Smelting, reference will be made to the proposals which have been made to carry on the reduction of  $\text{PbS}$  to metal by means of an air blast.

**Lead Chloride**,  $\text{PbCl}_2$ , crystallises in rhombic white silky needles, having a specific gravity of 5·8, soluble at  $12^\circ \text{C}$ . in 135 and at  $100^\circ \text{C}$ . in less than 30 parts of water, from which solution it is partly precipitated on addition of free  $\text{HCl}$ . It fuses below a red heat, and without volatilisation, to a mobile liquid, which sets on cooling to a horny, translucent mass.

The electrolysis of molten lead chloride has been suggested as a means of refining lead, but the method has never been carried out in practice.

**Lead Silicates.**—Lead oxide and silica begin to combine at the softening temperature of the oxide—i.e., below  $800^\circ \text{C}$ .; and if both substances are finely divided, and the heat maintained at this temperature, combination will be effected without the mixture becoming fluid. According to Percy, all the silicates of lead up to the trisilicate,  $2\text{PbO} \cdot 3\text{SiO}_2$ , are readily fusible at low temperatures, the monosilicate in particular,  $2\text{PbO} \cdot \text{SiO}_2$ , being as fluid as water. Higher silicates are successively less fusible up to  $2\text{PbO} \cdot 9\text{SiO}_2$ , which only melts to a substance resembling porcelain, while  $\text{PbO} \cdot 18\text{SiO}_2$ , can barely be fritted at the highest temperature obtainable. All the fused silicates of lead are yellow, the presence of other metallic oxides altering the colour, as seen in the scorification process of silver assaying.

According to recent investigations of Mostowitsch,§ no definite compounds are formed when  $\text{PbO}$  and  $\text{SiO}_2$  are melted together, but only solutions, which in the liquid state when the oxygen ratio of  $\text{PbO} : \text{SiO}_2$  exceeds that of 1 : 2 dissolve  $\text{PbO}$ , the amount dissolved increasing with the temperature and the excess falling out of solution as the temperature falls. Mixtures of  $\text{PbO}$  and  $\text{SiO}_2$  become fluid mostly below  $750^\circ$ , all those containing more than  $2\frac{1}{2}$  equivalents of  $\text{PbO}$  to  $1\text{SiO}_2$  become as fluid as water below  $800^\circ \text{C}$ .,  $2\text{PbO}$  to  $1\text{SiO}_2$  requires  $850^\circ$ ,  $4\text{PbO} : 3\text{SiO}_2$  over  $880^\circ$ , and  $\text{PbO} : \text{SiO}_2$  over  $940^\circ \text{C}$ . before they can be poured. All of these silicates are readily soluble in dilute acetic acid or ammonia or other alkali, and even water attacks them.

\* *Ibid.*, p. 217.

† *Comptes Rendus*, 1895, vol. cxx., pp. 1164-1167; and abstract in *Journ. Soc. Chem. Ind.*, vol. xiv., p. 807.

‡ *Journ. Chem. Soc.*, 1898.

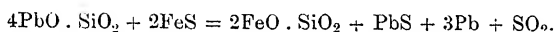
§ *Metallurgie*, 1907, iv., p. 648.



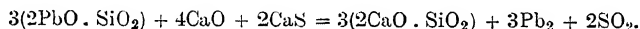
Reduction of these silicates by a current of hydrogen, first to  $\text{Pb}_2\text{O}$  and then to  $\text{Pb}$ , commences at temperatures of from  $350^\circ$  to  $600^\circ$ , while reduction with  $\text{CO}$  or  $\text{CH}_4$  requires temperatures of  $500^\circ$  upwards.

Silicates of lead are not readily decomposed by carbon or sulphur, but are decomposed by carbon in combination with lime, or some other base which combines with the liberated silica. Iron in excess effects a complete decomposition of all lead silicates at a bright red heat, forming monosilicate,  $2\text{FeO} \cdot \text{SiO}_2$ .

Sulphides of iron, baryta, and lime all decompose basic or subsilicates of lead at comparatively low temperatures, forming silicate of the base, sulphurous acid, and metallic lead. The reaction in the case of iron sulphide is as follows:—



The monosilicate is only partially decomposed, and requires for complete decomposition a higher temperature and the addition of a free base, such as lime:—



The bisilicate and lower silicates are readily decomposed by nitric acid, but decomposition is not complete in the case of the trisilicate, and higher silicates are attacked with increasing difficulty.

**Lead Fluosilicate**,  $\text{PbSiF}_6$ , crystallises in brilliant crystals, like those of lead nitrate, containing four molecules of water of crystallisation. It is readily soluble at  $15^\circ \text{C}$ ., only 28 per cent. of its weight of water being required to form a syrupy solution of 2.38 specific gravity, while at  $60^\circ \text{C}$ . the salt melts in its own water of crystallisation. Dilute neutral solutions are partially decomposed on heating with formation of a basic salt, but the decomposition is arrested if the solution contain 2 per cent. of free acid. A solution containing about 8 per cent. of  $\text{PbSiF}_6$  and 11 per cent. of free  $\text{H}_2\text{SiF}_6$  is used as electrolyte in the electrolytic refining of lead (*q.v.* Chap. xviii.).

**Lead Borates**.—Lead oxide and boric trioxide melt together in all proportions, and the compounds formed are more fusible than the corresponding silicates. Advantage is taken of this fact in lead and silver assaying.

The *haloid* compounds of lead are very volatile, and form volatile chloro-sulphides and chloro-phosphates. They are readily reduced by contact with carbon and metallic bases at a red heat.

**Lead Poisoning**.—Of the compounds of lead, all those soluble in water or in dilute acids are exceedingly poisonous, whereas the insoluble compounds appear to be innocuous. The effect upon the body is cumulative, small quantities once absorbed not being readily eliminated. The oxide and carbonate of lead appear to be poisonous, chiefly because, when breathed or taken into the stomach with food, they are dissolved by the gastric juices, and, after causing colic, are carried into the circulation, where they are deposited in the bones, and affect the nervous system, causing paralysis. Exposure of men to breathing lead fumes will cause poisoning, but in modern works the draught is usually so good as to minimise the danger in this direction. The idea formerly held that in ordinary lead works that metal was absorbed through the skin, as occasionally happens to potters and glaziers, is exploded, and it is now generally believed that only the mouth and nose

required to be guarded by good ventilation to carry away fumes, by using respirators in dangerous situations, such as cleaning flues and repair work inside hot flues and furnaces, and by strict cleanliness everywhere to prevent fume or dust coming into contact with food or drink, including cleanliness in person on the part of the workmen themselves.

At most lead works plain or oatmeal water acidulated with sulphuric acid is provided for the men to drink, with the idea of converting into insoluble sulphate any lead taken into the system, while personal cleanliness is advised, and as far as possible enforced. Hot baths are provided, and the men are encouraged to use them. At one works in Spain the men are encouraged to take a dose of Epsom salts once a week, and a hot bath at least as frequently, note being kept of those who do not conform to the customs. Should any such recalcitrant employé become "leaded"—i.e., affected with lead colic—he is promptly "laid off." Hutchings describes\* as efficacious the use of a lozenge made of sulphur, sugar, and peppermint, with  $\frac{1}{2}$  grain of sodium sulphide, which serves to convert any lead absorbed into harmless sulphide.

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\* Blum and Hutchings, "Untersuchungen über bleivergiftungen und ihrer Verhütung im industriellen Betrieb," Frankfort a. M., 1900, quoted in *Min. Ind.*, vol. x., 1901, p. 437.

## CHAPTER II.

## LEAD ORES.

METALLIC lead itself is found in nature, but only as a mineralogical curiosity. The principal minerals containing lead are the following :—

**Galena**,  $\text{PbS}$ ; 86.6 per cent.  $\text{Pb}$ .—This, the commonest of lead minerals, is frequently found in groups of cubical or octahedral crystals; also massive, filling cavities. It is found in every country in the world, and in almost every variety of deposit, from true fissure-veins to impregnations in certain beds of limestone, dolomite, and sandstone. In most cases it is so mixed with other vein or rock matter, as to require some process of “dressing” to fit it for treatment by the smelter; and very often the admixture with other metallic sulphides is so intimate as to give rise to great loss in these preliminary mechanical operations.\* The subject of ore-dressing scarcely appertains to metallurgy, although it is slightly touched upon in the chapter on the Treatment of Zinc-lead Sulphides, and the student may consult with advantage the compendium given by Foster in his *Ore and Stone Mining*,† and for further details the exhaustive work on *Ore-Dressing* by Richards.

Galena is almost always argentiferous, and its silver contents may be present either as isomorphous sulphide,  $\text{Ag}_2\text{S}$ , or as some distinct silver-bearing mineral (e.g., argentite, stephanite, polybasite, fahlerz) finely disseminated through the cleavage planes; this difference is important as regards the dressing operations. In the former case the loss of silver in dressing is proportionate to the loss of lead; in the latter and by far the commoner case, the loss of silver in dressing greatly exceeds the loss of lead; since the silver-bearing mineral, besides being finer to begin with, is also both lighter and much more brittle, so that it is driven into the finest slime, which floats away in and upon the surface of the dressing water. A notable case of this kind is to be found in the Zeehan and Dundas field of Tasmania, where, according to a series of assays of ores from different mines and workings given by Thomae,‡ the hand-picked ore contains on an average 94 ozs. silver per ton for 66 per cent. of lead, while the dressed ore carries only 76 ozs. silver for 70 per cent. lead, a clear loss of 25 per cent. of the silver in the fine slimes carried off by the dressing water. Another instance of the same fact is to be found in the fine “float” or scum from the jigs treating Broken Hill ores, which frequently assays from 60 to 80 ozs. silver, although the lead

\* See Chap. xvi. for the losses in dressing Broken Hill ores.

† Reference may also be made to Kunhardt, *Practice of Ore Dressing in Europe*, New York, 1892; to Bellom, “On the Mechanical Preparation of Ores in Germany,” in *Ann. des Mines*, 1891, vol. xx., p. 1; and to two papers in the *Proc. Civ. Eng.*, viz., Curtis on “Gold Quartz Reduction,” and Commans on the “Concentration and Sizing of Crushed Minerals” (*Proc. Civ. Eng.*, vol. cviii., p. 97, and vol. cxvi., p. 3, respectively).

‡ *Trans. Inst. Min. Met.*, vol. iv., p. 60.

concentrates produced in the same jigs run only from 25 to 30 ozs. The poorest galena as regards silver contents is that from Bleiberg (Carinthia), which contains only 1 dwt. per ton of ore; the richest known is that from Schemnitz (Hungary), a specimen of which, according to Beudant, gave 7 per cent. of silver, or over 2,200 ozs. per ton.\*

It is a common belief among miners that the grain of galena is influenced by its richness in silver, the general idea being that a coarse-grained galena is likely to be poor in silver, and a fine-grained variety rich. No general rule of the kind can be formulated;† nevertheless, as regards certain districts, some connection is observable between the grain of the galena and its silver contents. For instance, at Broken Hill, N.S.W., the coarse-grained galenas are said to be generally richer in silver than the fine-grained; while, according to the author's experience in various parts of Mexico, the contrary rule often prevails as regards the ores of each particular mine, though the coarse-grained galenas from one mineral deposit may be richer than the finest-grained varieties from another.

Gold itself may possibly occur isomorphously replacing part of the lead in galena, for Nevins describes‡ a galena containing 268 ozs. Ag and 15 ozs. Au per ton, which failed to show any native gold by panning. The author has had a similar experience in Mexico.§

**Cerussite**,  $\text{PbCO}_3$ ; 77·5 per cent. Pb.—This mineral is seldom found pure in quantity, except when occurring, as at Leadville and Broken Hill, in secondary crystals distributed through the "gozzan" of large complex ore bodies. In these cases, though itself comparatively free from silver, it is found associated with native silver and its haloid compounds, especially the chloride and chloro-bromide. It is nearly always mixed with more or less anglesite, and with oxide of iron and other vein material. Important deposits at Laurium (Greece), Monteponi (Sardinia), Eureka (Nev.), Leadville (Colo.), Sierra Mojada and Catorce (Mex.), and Broken Hill (N.S.W.), are now approaching exhaustion, but other important localities which still yield large quantities of this ore are Carthagera and Almeria (Spain), St. Eulalia and Velardeña (Mexico).

**Anglesite**,  $\text{PbSO}_4$ ; 68·3 per cent. Pb.—Like the preceding, this mineral is invariably a secondary or decomposition product of galena, and it is found mixed with the carbonate everywhere, generally, however, in small proportion. When crystallised, it is, like the carbonate, nearly free from silver. The ores of Eureka (Nev.) and Hornsilver (Utah) were particularly rich in anglesite.

**Pyromorphite**,  $3\text{Pb}_3(\text{PO}_4)_2 + \text{PbCl}_2$ , with 69·5 per cent. Pb, and **Mimetesite**,  $3\text{Pb}_3(\text{AsO}_4)_2 + \text{PbCl}_2$ , with 76·2 per cent. Pb.—These two minerals are mentioned together, for, although the formulæ of pure specimens are as above, the former frequently contains  $\text{As}_2\text{O}_5$ , and the latter large quantities of  $\text{P}_2\text{O}_5$ ; while both may contain lime replacing the lead, and fluorine replacing the chlorine. Both are comparatively free from silver, and occur associated with cerussite in the upper decomposed portions of lead lodes, pyromorphite being, perhaps, on the whole, the commoner of the two, although mimetesite,

\* Phillips, *Elements of Metallurgy*, 1891, p. 617.

† v. Percy, *Metallurgy of Lead*, p. 97.

‡ *E. and M. J.*, 1905, lxxx., p. 769.

§ *Mineralogical Magazine*, vol. xiii., No. 62, p. 362.

when it occurs at all, is often more abundant. The ores of Eureka (Nev.) contained much arseniate.

**Vanadinite**,  $3\text{Pb}_3(\text{VO}_4)_2 + \text{PbCl}_2$ , occurs in considerable quantities at the Santa Marta mines in Murcia (Spain); but although an important source of vanadium, it cannot be said to have any commercial importance as an ore of lead.

Other lead minerals\* are *Crocoisite*,  $\text{PbCrO}_4$ ; *Wulfenite*,  $\text{PbMoO}_4$ ; *Stolzite*,  $\text{PbWO}_4$ ; *Mendipite*,  $\text{PbCl}_2 \cdot 2\text{PbO}$ ; *Lanarkite*, *Phosgenite*, *Clausthalite*, &c., as well as sulphantimonides, such as *Jamesonite*, *Bournonite*, &c. Seeing, however, that none of these minerals occur in large quantities, or otherwise than associated with other ores of lead, they cannot be said to be of any metallurgical importance.

**Commercial Lead Ores.**—For metallurgical purposes the ores of lead are divided into "sulphide ores," consisting in the main of galena, with greater or less admixture of blende, pyrites, chalcopyrite, mispickel, fahlerz, &c., and "oxidised ores," called in America "carbonates," under which general term is included not only cerussite itself, but all its admixtures with anglesite, pyromorphite, &c.

**Oxidised Ores.**—It will be understood that the oxidised ores are not essentially a product of different districts from those producing sulphides, but that they are merely the surface gozzans, into which large masses of the latter have been decomposed by atmospheric agencies; anglesite representing an intermediate stage in the transformation of galena into cerussite. The change from oxidised ores to sulphides may take place gradually, as at Catorce and some other Mexican mining districts, or suddenly, as on Iron Hill, Leadville.† The depth at which the change takes place also varies considerably; thus at Broken Hill (N.S.W.) large masses of unoxidised sulphides are found at only 200 feet from the surface, while at Eureka (Nev.) oxidised ores continued down to 1,300 feet.‡ The principal factors which determine the depth to which oxidation extends are—(1) Depth below surface of permanent water level of country; (2) permeability of the rocks; and (3) amount of rainfall.§ By atmospheric oxidation the pyrites of the original mixed sulphide ore becomes iron oxide in the gozzan, chalcopyrite forms oxide of iron and carbonate of copper, blende is partly leached away as zinc sulphate, but a part remains as calamine and smithsonite. The silver contents of the original sulphides partly remain as sulphide or sulphantimonite, partly become native silver, but more generally form haloid compounds with the small quantities of haloid elements contained in the air and meteoric water. In this way the silver of the Leadville "carbonates" was chiefly found as chloride, while at Broken Hill the chlorobromide, bromide, and iodide, and in Chili the iodide and bromide are most commonly met with. Carbonate ores are generally richer in silver than the deep-seated sulphides, to the oxidation of which they owe their origin. This may be due to several causes, but the principal one is undoubtedly the much greater solubility of lead

\* For details, works on mineralogy should be consulted, especially Dana, *System of Mineralogy*, New York, 1892.

† Blow, *Trans. A.I.M.E.*, vol. xviii., p. 170.

‡ Curtis, "Silver Lead Deposits of Eureka," *Monograph VII. U.S. Geol. Survey*.

§ For further information on this interesting subject the student is referred to a paper by Halse on "Deep Mining in Mexico," *Trans. Inst. Min. Met.*, vol. iii., Part III., p. 418.

sulphate and carbonate, than of silver chloride and sulphide. Small nodules of galena are frequently found in carbonate ores, which are very much richer in silver than the surrounding matrix, but these are of secondary origin in most cases. Any gold contained in the original sulphides remains in the carbonates in the native condition.

Carbonate and other oxidised ores can be but seldom concentrated to advantage, owing to the comparatively low specific gravity of the oxidised lead minerals compared with galena, and to their great friability compared with the quartz, limonite, and other gangue. Concentration of carbonate ores, however, has been successfully practised at Tombstone (Ariz.),\* and more recently at the British Broken Hill Co.'s Works at Broken Hill (N.S.W.). The loss of silver in the fine slimes is usually more serious than the loss of lead. Attempts have been made to leach out part of the silver contents with a hyposulphite solution before concentrating for lead, also to separate most of the lead by a dry concentration process prior to working up the tailings by amalgamation or lixiviation processes; but neither attempt can be said to have been very successful.

**Sulphide Ores.**—With reference to these ores, it may be said that, as a general rule, those deposits in which the galena occurs most free from other metallic sulphides will be poorer in silver than those in which it is intimately associated with pyrites, blende, and other sulphides. Deposits in sedimentary rocks are almost always poorer in silver than fissure vein deposits associated with metamorphic or eruptive rocks.

Galena occurs comparatively free from both blende and pyrites at Engis (Belgium), Mechernich (Rhen.-Pruss.), and in the upper parts of the limestone deposits (Lower Silurian) of Wisconsin and Illinois. Galena associated with blende alone is found at Raibl and Bleiberg (Carinthia) and at Joplin (Mo.), while in the mines of St. Joe and La Motte (Mo.) it is associated with pyrites to the exclusion of blende. In all these cases the galena is almost free from silver, the ratios of that metal to lead varying, according to a useful little table given by Hofman,† from 1 dwt. Ag for 70 per cent. Pb (Bleiberg, Carinthia), to 1½ ozs. Ag for 80 per cent. Pb (Granby, Mo.), and 4 ozs. Ag for 60 per cent. Pb at Mechernich. All the above are distinctly sedimentary deposits, either in sandstone, as at Mechernich, or in limestone, as at the remaining localities cited. The Spanish galena deposits (Linares, Almeria, &c.), which are also comparatively free from other metallic sulphides, are of an intermediate character, averaging, as prepared for smelting, from 6 to 10 ozs. Ag for 75 per cent. Pb.

When galena is found most intermingled with other sulphides (especially with complex sulphides and sulphantimonides like fahlerz), and more particularly when in contact or in connection with crystalline, metamorphic, or eruptive rocks, as in Bohemia, Saxony, Hungary, the Harz and the Rocky Mountains, it is usually richest in silver. As already pointed out, however, a large part of the silver contents of these complex sulphide mixtures does not exist as a constituent of the galena itself, but rather in distinct silver minerals, such as argentite, stephanite, and fahlerz, either finely disseminated through the cleavage planes of the galena, or through the matrix. Even the blende and pyrites in these mixed ores usually carry appreciable quantities

\* Church, *Trans. A.I.M.E.*, vol. xv., p. 601.

† *Metallurgy of Lead*, N.Y., 1893, p. 26.

of silver; in some parts of the great Broken Hill Proprietary Mine the blende carries nearly as much silver as the galena. In the Linares district of Spain galena is usually found almost free from admixture with blende and other metallic sulphides, and it usually carries only 6 to 12 ozs. per ton. The author has, however, in his possession a specimen of blende from the Tortilla Mine in that district which contains 100 ozs. silver per ton, although the galena from that same mine is very regular in its content of only 12 ozs. silver per ton. In another interesting case described by Freeland,\* some ore from the Minnie Mine at Leadville was separated as nearly as possible into its component parts of galena, blende, and pyrites, and the blende turned out to be much richer than the galena. The following analyses are by Boggs:—

	Galena.	Blende.	Pyrites.	Mixed.
Pb, . . . . .	72·65	6·71	2·21	50·86
Zn, . . . . .	5·66	55·08	14·24	12·86
Fe, . . . . .	1·60	4·00	35·40	9·30
S, . . . . .	15·56	32·44	44·76	24·50
Ag, . . . . .	0·14	0·324	0·014	0·039
Residue, . . . . .	4·12	0·92	2·70	1·88
	99·83	99·474	99·324	99·439
Silver, oz. per ton, . . .	41½	94½	4½	11½

This is, of course, rather an exceptional case, and by no means the rule in the Leadville district; thus, Taylor and Brunton report † a similar series of analyses from the Col. Sellars Mine, in which the galena contains 20 ozs., the pyrites 5 ozs., and the blende only 3 ozs. The series of analyses mentioned has been quoted in full merely as a warning against the too common assumption that in dealing with low grade complex ores the only thing to aim at is a successful concentration of the lead. It may frequently happen that in spite of a fairly good saving of this metal, what with the losses of silver in blende and pyrites tailings, and what with the loss of finely divided true silver minerals in the slimes, the lead concentrates do not contain more than one-half of the silver value of the original ore. It is, therefore, in many cases preferable to smelt the whole mass of a complex ore body rather than to make any attempt at concentration. Analyses of smelting ores will be given in the sections dealing with the various processes of treatment.

\* "Iron Hill Sulphide Deposits," *Trans. A.I.M.E.*, vol. xiv., p. 181.

† *E. and M. J.*, May 8, 1886.

## SECTION II.—LEAD SMELTING.

### INTRODUCTORY.

PRACTICALLY speaking, only smelting methods are applicable to the extraction of lead from its ores. Wet methods, based on the leaching out of  $\text{PbCl}_2$  or of  $\text{PbSO}_4$  by means of suitable solvents, and precipitation with iron, have been at various times suggested, but none have proved sufficiently promising to be worked on a commercial scale.

As already seen, by far the commonest lead ore is galena; comparatively small proportions only of carbonate, sulphate, and other ores are now accessible,\* and, as a rule, only as incidental accompaniments of large bodies of galena. The treatment of these oxidised ores, moreover, offers no special difficulty and requires no special appliances; they may be treated generally as roasted sulphide ores, oxidation having been effected slowly by atmospheric agencies instead of quickly in a furnace. The smelting of these oxidised ores in Australia, Mexico, and elsewhere will be described in its proper place, and the treatment of lead ores may be considered generally as referring only to galenas.

**Classification of Methods.**—Three principal methods of treatment may be distinguished :—

1. The *Roast and Reaction Method*, in which the galena is first roasted so as to convert part of it into  $\text{PbO}$  and  $\text{PbSO}_4$ ; the temperature is then raised so that these compounds react upon the unaltered sulphide with production of metallic lead and  $\text{SO}_2$ .†

2. The *Roast and Reduction Method*, in which the galena is roasted pretty completely to lead oxide, which is then reduced to metal with coke or charcoal. A small proportion of sulphate formed in the roasting is either transformed into silicate at the end of the roasting process, and then reduced in the smelting furnace with the help of iron oxide, or it may be reduced to sulphide in the smelting furnace, and so form a “matte” from which part of the metallic lead is again set free by iron. In any case nearly the whole of the lead is recovered direct in a metallic condition.

3. The *Precipitation Method*, in which galena is decomposed at a high temperature by the agency of metallic iron, forming metallic lead and a mixture of iron and lead sulphides, the so-called “lead matte.”

These methods are rarely employed quite independently of each other, for the residues of the first method are almost always worked up by the second or third methods, and the second method is almost always combined with the third, and often with the first also.

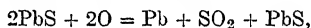
\* Except, perhaps, in parts of Mexico.

† For equations representing these reactions *v.* Chap. i.



Besides these three established methods, there are others which it will be convenient to mention here, though they have not yet passed the experimental stage.

A fourth method, which has been proposed, is that of direct oxidation by means of a blast of air without fuel, but it has not yet got beyond the experimental stage. According to F. Laur,\* Germot and subsequently Catelin at Clichy (France) have experimented on the direct production of metallic lead through the oxidation of molten galena by a blast of air. The reaction which takes place is the following :—



One-half of the lead is reduced, carrying with it practically all the silver, and the heat produced is, in the case of rich pure galenas, nearly sufficient to volatilise the other half of the lead as sulphide (black fume). The furnace employed is a circular cupola or shaft about 21 inches diameter and 15 feet high, closed at the top, except for charging tubes and a gas delivery tube, through which the gas and fume are exhausted by means of a fan between the condensing chamber and the stack, also a tuyere tube suspended in the centre, and capable of being lowered or raised at will. The furnace having been heated up, and 100 kgs. of lead and 200 kgs. of melted galena having been poured in from suitable tilting melting furnaces, the tuyere is lowered into the galena, and blast is turned on at the rate of about 350 cubic feet per minute, after which from 1 to 3 tons of ore is gradually added through the charging tube. If the blast is properly adjusted, nothing but black fumes of PbS pass to the condensing chamber; if, however, the fume becomes light-coloured the blast is turned off and lead is tapped from the crucible. When matte begins to follow the lead, the tuyere is lowered and air again blown through till the slag becomes liquid through scorification of the remaining lead, when it is tapped off, and the cycle of operations again begun.

The process has not passed the experimental stage, and it is perhaps not likely that it ever will do so, but it is, nevertheless, of great theoretical interest.

Another method, which is said † to be actually at work, is that of direct electrolytic reduction. Finely crushed galena is spread in a layer 1 inch deep on the bottom of flat, conical, hard lead pans, covered on the upper part of their sides by a thin coating of india-rubber, the charge of each pan being 3 lbs. Eleven pans or cells are made into a pile, the bottom of each pan forming the anode, while the layers of galena form cathodes. Each pile of cells is electrolysed in five days by a current of 33 amps. at 2.9 volts, four piles are connected in series, and any number of the groups of four are run in parallel. The lead takes a spongy form, and swells to a thickness of 3 to 4 inches, the sulphur liberated as sulphuretted hydrogen passes away with excess of hydrogen, and is burned. The lead is suitable for use in storage batteries after washing and drying, or it may be converted into any of the commercial compounds of the metal.

As regards the three established methods, the choice of method depends partly upon the quantity and nature of the impurities in the ore, partly

\* *Oesterr. Zeitschr. f. B. u. II. Wesen*, L. xl., 55, Oct. 4, 1902. Abstracted in *E. and M. J.*, Nov. 1, 1902, p. 574.

† *Electrochemical Industry*, 1902, i., p. 18, and *Min. Ind.*, xi., 1903, p. 450.

upon local conditions, such as the kind of fuel available, and the price of labour.

The *Roast and Reaction Method* may be carried on in hearths or in reverberatories, and requires raw fuel only, but is suited only to high grade galenas containing 60 to 70 per cent. lead and upwards, since large quantities of other substances interfere with the reactions of  $PbO$  and  $PbSO_4$  on  $PbS$ . It is quite inapplicable to ores containing much quartz or silicates, which would form a fusible lead silicate, and so put a stop to the reaction. The advantages of this method for dealing with rich ores are:—(1) Yield of a very pure lead; (2) use of cheap forms of fuel; (3) no fluxes required; (4) comparatively small volatilisation loss. Its disadvantages are:—(1) Production of a very rich slag which must be re-treated; (2) heavy labour cost; (3) large quantity of fuel required.

The *Precipitation* method in its original form was carried out either in reverberatories or in blast furnaces, and involved actual additions of metallic iron to the charge in the form of scrap. This form of precipitation is still to be seen in some localities in connection with the reduction of grey slags and other oxidised products in blast furnaces, and with the reduction of litharge and similar refinery bye-products in reverberatories. Instead of adding metallic iron to the charge, however, this metal can be formed in the furnace by the use of coke fuel, and much ferruginous flux. In connection with ore-smelting the method only survives now as a blast-furnace method in this form.

It is only applicable to rich lead ores containing comparatively small quantities of antimony, arsenic, &c., because these metals are also reduced by iron, rendering the lead very impure. Silica and other non-metallic minerals do not interfere with the process; and copper ores are well suited to it because the whole of that metal, even if existing in very small proportions, becomes concentrated together with part of the silver in the matte, from which both can be recovered. The advantages of this process are that it dispenses with a preliminary ore roasting, and gives ultimately a good extraction of lead with comparatively low volatilisation losses of both lead and silver. It has, however, many grave disadvantages, especially the high consumption of coked fuel, and the slow, because only partial and gradual, recovery of the lead, together with a complicated matte treatment which makes the method a costly one for labour.

As a distinct process for the smelting of ores, therefore, the method is practically obsolete, surviving only in the Upper Harz at Clausthal under exceptional local conditions. Some precipitation, however, undoubtedly comes into play in ordinary blast-furnace smelting whenever reduction is good; and the regular addition of tin-cans and other sheet-iron scrap to the furnace charge, in order to improve reduction, has only been extinct in leading American works a comparatively few years, while it is still occasionally employed as a remedy for exceptional furnace conditions.

The *Roast and Reduction Method* is applicable to all kinds of ore, but especially to the poor siliceous, pyritous, and blendiferous ores which are so common, and which cannot be treated by the foregoing methods. It comprises two independent processes, carried on in separate furnaces. The roasting is conducted in the various ways to be described, and the smelting of the roasted ore takes place in blast furnaces, the construction

of which shows great variation in detail, though the principle is always the same.

Lead having such a great affinity for silver, which it dissolves readily in all proportions, poor lead ores are to a large extent smelted with a variety of ores of silver and gold; the object being to utilise the lead contents in the charge-mixture as a solvent or medium of concentration for the precious metals present, the commercial value of which is greater than that of the lead itself. Modern blast-furnace lead smelting, therefore, in a majority of cases, is to an even greater extent silver smelting; but in view of the fact that lead cannot be lost without at the same time losing silver, and that a good recovery of the former means, generally speaking, a good recovery of the latter, we may deal with the smelting of ores of the precious metals upon a lead basis, as if it were lead smelting pure and simple.

The roast and reaction method may be carried out either in reverberatories or in hearths, the residues in either case being run down in blast furnaces. Whatever the appliance employed, only a certain proportion of the total lead contents of the ore can be recovered direct in the first instance, together with almost all of the silver, the proportion varying from 80 to about 84 per cent. as a maximum. A good part of the remaining lead is locked up in the residues referred to; but, in addition, whatever the appliance employed, there is always considerable loss of lead by volatilisation, which becomes of greater importance in proportion as the ores are richer in silver. In all cases, therefore, the furnace flues are connected with a more or less complete system of flues and condensing chambers for recovering the flue dust. A separate chapter is devoted to this subject, which need not, therefore, be further dealt with here.

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## CHAPTER III.

## LEAD SMELTING IN REVERBERATORIES.

**Preliminary.**—The ore for this process must be crushed sufficiently fine to pass a 4 or 5-mesh sieve, an operation which is usually performed before the ore reaches the smelting works, as part of the “dressing” or “concentration” almost always required to produce ore of sufficiently high grade for the process. In all cases the process comprises two stages, which follow each other in the same furnace, and which are rapidly alternated several times.

*1st Stage or Roasting.*—The ore is spread in a layer of from 3 to 4 inches over the hearth of the furnace, and gradually brought to a low red heat, say  $500^{\circ}$  to  $600^{\circ}$  C., at which it is kept with free access of air, being continually raked to expose new surfaces and to prevent clotting, which would suspend the process of oxidation. The roasting is not allowed to complete itself, sufficient undecomposed sulphide being left to bring about the reactions described in Chap. i. The rabbling in of small quantities of lead carbonate and sulphate ores or of litharge is of great advantage, since it shortens the period of roasting. Could a sufficient amount of these oxidised substances be obtained to mix with the galena in proper proportion, roasting might, in fact, be dispensed with altogether, and the reaction period entered upon at once in a hot furnace; but this is rarely, if ever, possible.

*2nd Stage or Reaction.*—The temperature is raised so that the oxidised compounds may react upon unaltered sulphide according to the equations given in Chap. i. The reactions take place best at the temperature when the charge begins to soften, and are incomplete if it be allowed to melt. The charge is worked up with the rabble, lime being generally added to stiffen it, and the cooling consequent on opening the doors stops the first reaction period, and brings about a second roasting, followed, when the doors are closed and the temperature again rises, by a second reaction. These operations are repeated until there is not enough lead sulphide present to react upon the sulphate and oxide. The last operation, then, consists of adding fine charcoal or coal, which is well rabbled into the charge with the object of reducing part of the residual lead oxide to metal.

The chief difficulty is to oxidise the charge uniformly, only just enough to secure the theoretical molecular proportions for the reaction; and as roasting goes on more or less during the whole period of reaction, it usually happens that considerable excess of oxide and sulphate is produced. This excess, together with more or less undecomposed sulphide, globules of metallic lead, and the whole of the other substances mixed with the galena, remains on the hearth in a half-melted condition as the so-called “grey slag,” which is raked out for separate treatment. In former times this

sometimes consisted merely of crushing and washing to save the metallic lead, but generally an attempt is made to get out the combined lead also. In some cases, especially with rich ores, the residues from which are small in amount, the slags are treated in the same furnace, coal dust being rabbled in and the temperature raised to secure the reduction of the combined lead; but in this case the loss by volatilisation is considerable, and a notable amount of lead still remains as sulphide and silicate in the final residues. It is far better, therefore, to leave a larger proportion of lead in the grey slag, and work this up afterwards in blast furnaces.

*Yield of Metal.*—The yield of metal depends chiefly upon the quantity and nature of the impurities present, but is also largely influenced by the hardness and mechanical condition of the ore. Comparatively coarse ore retains its granular character longer and agglomerates less readily than slimes or table concentrates, consequently gives a larger direct yield. Soft well-crystallised ores—particularly if readily decrepitating on the application of heat—also give a lower yield than the harder, fine-grained and less readily cleavable ores.

**Influence of Foreign Substances.**—*Quartz, clay,* and all other *silicates* are very injurious, as they form lead silicates which cannot be decomposed at the low temperatures prevailing in the furnace, besides glazing over particles of undecomposed sulphide, and so protecting them from the reactions. The practical limit of silica in ores which can be advantageously treated by the roast and reaction process is about 4 per cent.; but a very much smaller amount is sufficient to glaze particles of ore and so obstruct the reactions, unless the heat is kept very low.

*Calcite (limestone or dolomite),* like other inert substances, may, when in large amount, retard the reactions by interfering with the direct contact between the oxidised lead compounds and unaltered sulphide. Apart from this, however, it acts on the whole advantageously by hindering the fusion of the charge, at all events up to amounts of 10 or 12 per cent. When not present in the ore, lime is frequently added in small quantities during the process with the object of stiffening the charge and preventing fusion.

*Barytes* and *fluorspar* remain inert during the process, but when present together tend to increase the fusibility of the charge.

*Chalcybite* often occurs as a gangue of galena ores, but can be mostly removed by dressing. Small quantities which remain with the ore lose their  $\text{CO}_2$  during the roasting and become converted into magnetic oxide of iron, which assists the reaction by helping to stiffen the softened charge and, perhaps, also, by acting as an oxygen carrier.

*Pyrites* in small quantity has a decidedly favourable influence on both the roasting and reaction periods. During the former it becomes oxidised to iron sulphates, which subsequently give off  $\text{SO}_3$ , and so directly assist the formation of lead sulphate. During the latter, the ferric oxide formed tends to stiffen the charge, and probably also to act as an oxygen carrier. In quantities above 10 or 12 per cent., however, ferrous sulphide tends to form and run down prematurely as a matte or “regulus,” retaining, moreover, lead sulphide in the residues; and with 35 per cent. pyrites the roast and reaction process becomes not only uneconomical but impossible.\*

\* Bouly, *Ann. des Mines*, 1870, vol. xvii., p. 178.

*Chalcopyrite* behaves like pyrites, except that some copper is reduced with the lead.

*Blende* is partly converted into oxide and sulphate during the roasting and partly remains unaltered. Up to 4 or 5 per cent., the presence of blende assists the roasting process; but 10 or 12 per cent. retards the reactions very much; and 35 per cent., as in the case of pyrites, stops them altogether.

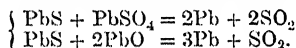
*Antimony* (stibnite, Jamesonite, Bournonite, zinkenite) is the most deleterious of all foreign elements, even in quantities of only 2 to 3 per cent. In the first place, both its oxide and sulphide are volatile, causing losses both of lead and silver. In the second place, they react upon each other, like the corresponding lead compounds, forming metallic antimony; part of this alloys with the lead, making it hard, and part volatilises again, carrying lead with it. Moreover, both oxide and sulphide are readily fusible, causing the ore to cake even during the roasting process; while the former combines with lead oxide to form lead antimoniate, which remains in the residues and causes them to yield a very hard lead when re-smelted.

*Arsenic* (mispickel, &c.), like antimony, is very detrimental, causing losses by volatilisation, and by combination with lead oxide to form arseniate. A portion also combines with the lead and contaminates it.

*Silver and gold* in the ores follow the lead reduced to metal, very little being left in the oxidised residue; the lead reduced first is by far the richest in precious metals, so much so that it frequently contains four or five times as much as the last lead, and seven to ten times as much as that in the residues.

**Classification of Processes.**—Until recently three main varieties of process have been distinguished—namely, the *Carinthian*, *English*, and *Silesian*. The first, however, is only a survival, and the last is but a trifling modification of the second. Besides these there was formerly in existence a distinct Cornish process (of which the so-called Breton process was a modification) now no longer in use.

The *Carinthian* process was only suited to pure rich ores, but from such extracted a large quantity of pure lead with a low loss, its disadvantages being the high cost for fuel and labour. It was characterised by small furnaces and charges, and by a low temperature throughout, with the object of producing as nearly as possible 1 mol. of  $\text{PbSO}_4$ , or 2 mols. of  $\text{PbO}$  for each mol. of unaltered  $\text{PbS}$ , so as to get the theoretical reactions:—



Coal was added to the residues in the same furnace so as to get out as much lead as possible.

This method was in use at Bleiberg, Carinthia, and also in Belgium; but has now given way\* to a modified Silesian process, followed by smelting the residues in Pils furnaces. There is, however, an American form of the process, which still exists to some small extent at isolated points in the Mississippi Valley.

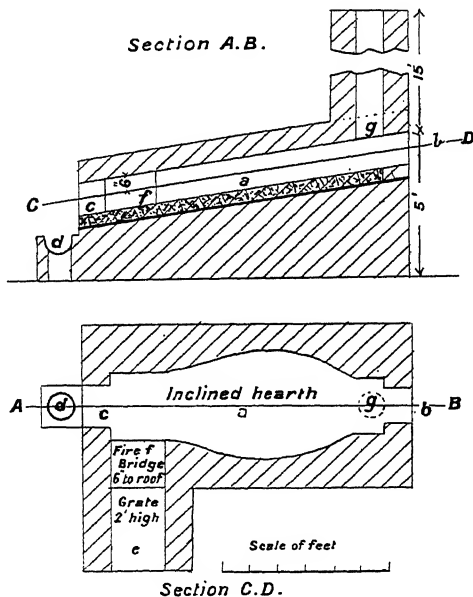
The *English* (*Flintshire*) process is conducted in large furnaces and at a high temperature, so as to get out as much lead as possible in the shortest time and with a small consumption of fuel; the volatilisation loss is, however, rather high. Owing to the high temperature, reaction is at first chiefly

\* Oesterr. Zeitschr. f. Berg- und Hüttenwesen, June 3, 1893, p. 283.

between  $\text{PbO}$  and  $\text{PbS}$ , but little  $\text{PbSO}_4$  being formed, except towards the end of the process.

The residues are treated in small blast furnaces (cupolas). The *Silesian process* is simply an improvement on the ordinary Flintshire method, with the object of saving labour and securing lower volatilisation losses. It is carried on in large furnaces, and at as low a temperature as possible; in consequence of which the proportion of lead obtained at first is smaller, and the residues for subsequent cupola treatment are much richer.

The extinct *Cornish process* was adapted to the treatment of 60 to 70 per cent. lead ores containing some copper and other impurities. The roasting and reaction stages were carried on in separate furnaces, and a larger quantity of  $\text{PbO}$  and  $\text{PbSO}_4$  was formed in the first stage than was required for



Figs. 1 and 2.—Carinthian Furnace, American Variety.  
(Vertical and Horizontal Section).

reaction upon  $\text{PbS}$  in the second stage. This excess of  $\text{PbO}$  was reduced, partly by the addition of "culm"\* and partly by scrap iron, the latter of which also served to collect copper and any remaining sulphur into a "regulus," which cleaned the slag so effectually that it could be thrown away. The loss by volatilisation was, however, heavy, owing to the high temperature employed, and the process was more expensive than the ordinary English process.

**The Carinthian Process.**—This process was, until recently, used at Bleiberg in Carinthia for treating galena concentrates of from 60 to 75 per cent. lead. A good description, with illustrations of the furnace adopted, is given by Percy.† The charges in this furnace were only 3 to 4 cwts., and

\* Anthracite slack.

† *Metallurgy of Lead*, 1870, pp. 261-271.

the slag from two successive charges was reduced in the same furnace with addition of charcoal.

An almost exactly similar process was largely in use in Missouri and Wisconsin for treating the rich lead ores of the Mississippi Valley, and still lingers in some places. A good description is given by Hofman.\* The furnace employed is in both cases essentially the same, and the American variety (which is the simpler) is shown in plan and section in Figs. 1 and 2. The charge in this furnace was about 1,500 lbs. of galena, which was worked off in twelve hours, extracting an unknown proportion of the lead present, and leaving a "grey slag" which contained 55 per cent. Pb, chiefly as oxide, but partly also as silicate and sulphate. The Carinthian furnace is now, practically speaking, extinct, except in a few isolated localities, having been replaced by other methods less costly for labour and less wasteful of fuel.

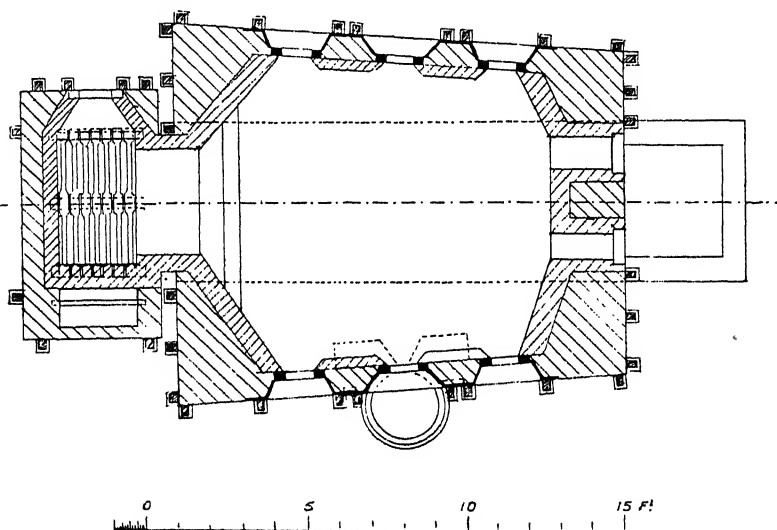


Fig. 3.—Flintshire Furnace.

**The English Method.**—As already mentioned, the characteristics of this method are the use of large furnaces and moderately heavy charges, which are worked off quickly at a high temperature, the direct yield of lead being high and the volatilisation loss also rather high. Two main varieties of the method may be described according to the style of furnace employed—viz., the *Flintshire*, and the Spanish reverberatory furnace or "*boliche*."

**The Flintshire Furnace.**—The ordinary form of this furnace is built upon an air vault, which serves to cool the hearth. In some modified furnaces, instead of this vault, the whole furnace is open underneath, and is supported on iron rails or bars; and in any case it must be very strongly bound with cast-iron plates, buckstaves, and tie-rods. As a rule, the tapping plate is made  $1\frac{1}{2}$  inches thick, while the plates in which the whole furnace is encased are  $\frac{5}{8}$  to  $\frac{3}{4}$  inch, preferably the latter. Common red brick is employed for all

\* *Metallurgy of Lead*, 1893, p. 89.



the outside work, for the lower bottom, and for all parts of the furnace not exposed to a high heat, but the interior lining is built of firebrick. The true

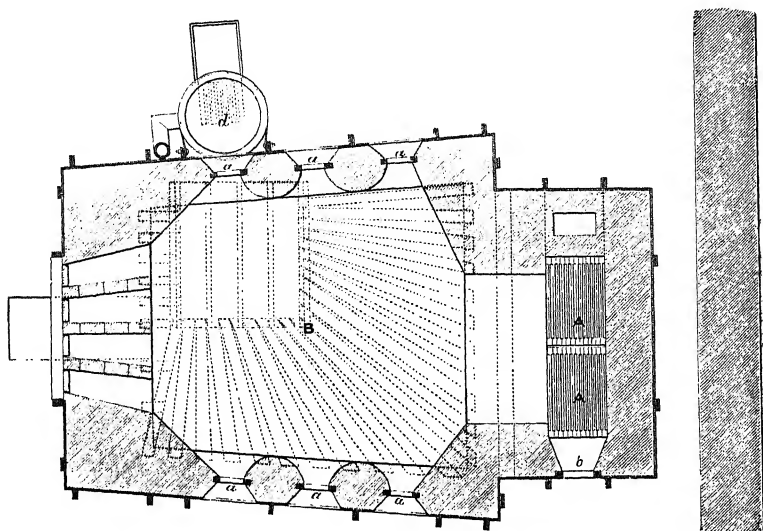


Fig. 4.—Flintshire Furnace, Couëron—Horizontal Section on C D (Fig. 5).

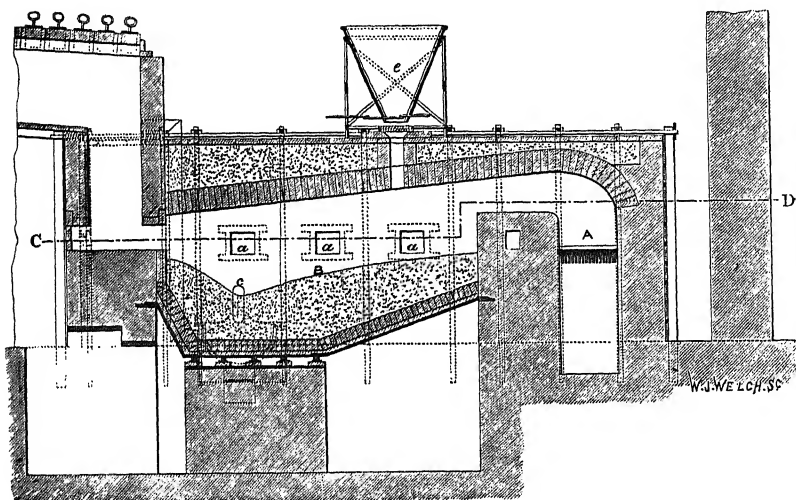


Fig. 5.—Flintshire Furnace, Couëron—Longitudinal Section.

working bottom of the furnace varies from 6 to 12 inches in thickness, and is composed of the grey slag produced in the smelting process, about 5 tons

being required for the formation of the bottom, which is patched as required with slags produced in the furnace itself. The grate is usually shallow, the ashpit being entirely open at the back, and frequently the ashes are allowed to drop into water, the steam from which perhaps has some beneficial effect in equalising the heat inside the furnace. Fig. 3\* shows the ordinary form of Flintshire furnace, in which the centres of the fireplace and flue are approximately in the longitudinal axis of the hearth, and the cast-iron lead well is situated in front of the middle working door at the front of the furnace. Figs. 4, 5, and 6† show Hutchison's modification introduced at Couëron, in which the line joining the grate and flue is set still further to one side of the longitudinal axis of the hearth, and the lead pot is placed in front of the third front door in order to withdraw the lead as far as possible from the influence of heat. This furnace also shows the construction on iron girders,

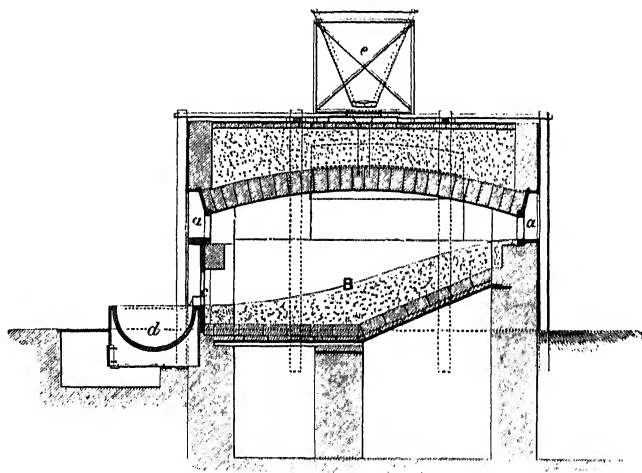


Fig. 6. — Flintshire Furnace, Couëron—Transverse Section through Taphole.

instead of the usual "air vault"; but these details are, of course, not essential. The firebridge is always air-cooled, and it will be noticed that the exit flue is divided so as to permit of the flame being more spread towards the back of the hearth (*i.e.*, the side opposite the well). The cast-iron pot is usually provided with a separate fireplace in order to keep the lead molten while it is being ladled. The door frames and skew backs are of cast iron,  $\frac{1}{2}$  inch thick.

**Mode of Working.**—The ores treated are from 75 to 80 per cent. by iron crucible assay. The usual charge is 21 cwts. dry weight (at Couëron, 27 cwts.), which, if not already fine enough, is crushed with rolls to about an 8-mesh; after which it is charged into the furnace, still at a red heat from the previous operation, and spread over the hearth.

**First Stage—"Calcination."**—The temperature is kept low, the damper being closed, and all the doors are left open so as to get as much oxidation

\* From Percy, *Metallurgy of Lead*, 1870, p. 225.

† From Phillips, *Elements of Metallurgy*, 1891, pp. 643, 644.

as possible. This stage lasts from an hour and a half to two hours by which time about one-third of the lead is oxidised—principally to  $\text{PbO}$ .

*Second Stage—"Roasting" or Reaction.*—The grate is freed from clinkers and fired up with coal, and the dampers are slightly opened, while the front doors are closed, and the charge is vigorously rabbled through the back doors. Effervescence takes place and much lead trickles down into the well. This stage lasts about half an hour, making a total of three hours from the charging.

*Third Stage—Melting down.*—The doors are closed, the damper opened and the fire urged so as to melt down the whole charge into the well. This stage lasts about half an hour (three and a half hours from charging).

*Fourth Stage—"Setting up."*—All the doors being thrown open to cool the charge, a shovelful of slaked lime is thrown upon the surface of the molten slag in the well, while the bed is well scraped with iron paddles to detach any adhering half-melted portions of the charge. All such material detached is well mixed by rabbling with that pushed back from the lead-well after thickening with lime, and the whole is then raked back and "set up" on the sloping part of the hearth immediately beyond the firebridge. This stage takes about half an hour (four hours from charging).

*Fifth Stage—Second Calcination.*—The damper being partly closed, the two doors near the fire are left open so as to oxidise the charge again during another period of about twenty minutes (four hours twenty minutes from charging).

*Sixth Stage—Second Roasting and Melting down.*—The damper is fully opened and all the doors closed. Vigorous effervescence with reduction of metallic lead first takes place, and then the remainder of the charge quickly melts down into the well on top of the lead. This stage takes half an hour to forty minutes (five hours from charging).

Two shovelfuls of lime being now thrown upon the surface of the slag and well rabbled in to thicken it, the slag is pushed back from the taphole and the furnace tapped into the pot. The slag is raked out through the middle front door and a new charge dropped and spread.

The above sequence of operations is varied at different works; thus with somewhat poorer ores it is common to subject the slag to a second "setting up," followed by a third calcination and melting, the whole process then taking six hours actual working time, or nearly seven hours including raking out the slag, cleaning the hearth and spreading a fresh charge. Usually, however, two charges are readily worked off in a twelve-hour shift by two smelters and a labourer.

The lead in the pot is allowed to cool before ladling, during which a considerable quantity of dross (consisting mainly of  $\text{PbS}$ ) separates, the separation being assisted by stirring in coal slack and hot cinders. The mixture of dross and slack is thrown back into the furnace-well, when a small quantity of lead immediately sweats out, and is allowed to run through the taphole before stopping this with a plug of clay or mortar for the next charge (already spread and "calcining").

**Analyses of Products.**—The following table adapted from Percy\* shows the actual condition of the charge at different stages of the process:—

TABLE III.—COMPOSITION OF CHARGE AT END OF VARIOUS STAGES.

Sample after :—	Ore charged.	First Stage. Calcination.	Third Stage. Melting.	Fourth Stage. Setting up.	Fifth Stage. Second Calcination.	Sixth Stage. Second Melting.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
PbS, . . . . .	89.95	63.82	53.32	24.76	4.35	0.90
PbSO <sub>4</sub> , . . . . .	...	3.42	4.78	6.94	14.02	9.85
PbO, . . . . .	5.15	27.25	31.49	43.12	47.50	48.87
ZnS, . . . . .	0.99	...	...	...	...	...
ZnO, . . . . .	...	1.25	3.00	5.02	5.72	7.52
CaO, . . . . .	0.65	1.32	2.14	7.38	11.51	12.68
Fe <sub>2</sub> O <sub>3</sub> and Al <sub>2</sub> O <sub>3</sub> , . . . . .	0.42	0.88	1.54	2.95	4.25	5.87
Combined SiO <sub>2</sub> , . . . . .	...	...	...	8.32	11.04	12.52
CO <sub>2</sub> , . . . . .	1.62	...	...	...	...	...
Insoluble residue, . . . . .	0.85	1.60	3.99	1.12	0.80	1.45
Total, . . . . .	99.63	99.54	100.26	99.61	99.19	99.66
Total Pb., . . . . .	82.71	91.36	78.66	66.22	47.86	52.88
Approx. weight of charge, . . . . .	21 cwts.	18 cwts.	8 cwts.	4 cwts.	3½ cwts.	2½ cwts.

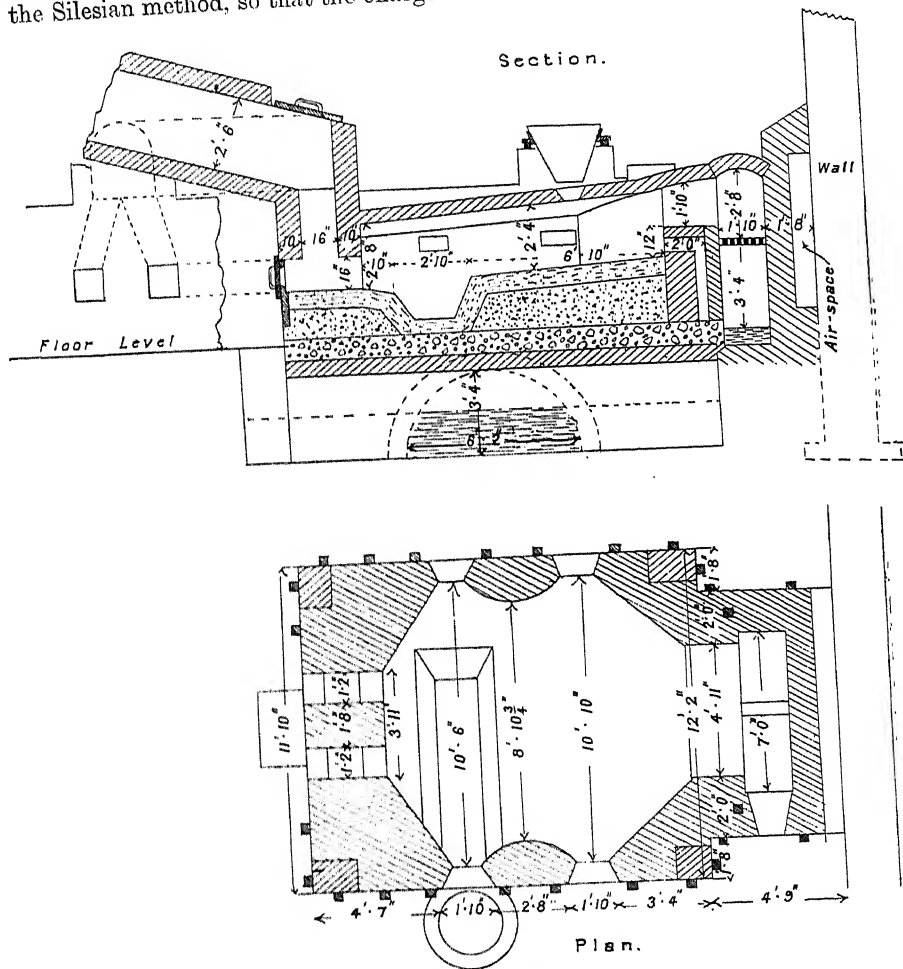
It is noteworthy that the proportion of PbSO<sub>4</sub> at first formed is comparatively small, the lead being chiefly converted direct into oxide. The temperature of the mass at the first melting down is so low that no lead silicate is formed. After the addition of lime and the elimination of more lead, however, the residue (which is now much richer in ZnO, Al<sub>2</sub>O<sub>3</sub>, and Fe<sub>2</sub>O<sub>3</sub>, as well as CaO) is so much more infusible that, at the increased temperature to which it is now exposed, PbO and SiO<sub>2</sub> enter into combination with each other. Percy discusses at some length\* the function performed by lime in the process, and comes to the conclusion that its action is essentially, if not exclusively, mechanical. While its principal function is doubtless mechanical, the author would suggest that possibly it may assist in liberating PbO from silicate formed during the first melting down, to act upon residual PbS; for, although the silicates of lead undoubtedly react upon PbS direct (as proved by Beck†), yet the temperature at which this reaction begins is in all probability above that which prevails at the second calcination stage of the lead smelting process. Percy himself shows, moreover, that at the temperature of the final melting down of grey slag there is no reaction between lead sulphate and silicate. The question of the function performed by lime in the roast reaction process will be considered more fully in the chapter on blast-roasting.

Seeing what a large excess of oxidised compounds is formed in the final stages of the process, it might be supposed that a better proportion of lead would be extracted by keeping the amount of oxidation low. It should be borne in mind, however, that besides the bad effect of a large quantity of PbS liquating out with the lead, a lessened oxidation at first would greatly increase the duration of the process, and, therefore, its cost; besides, in all

\* *Op. cit.*, p. 237.† Quoted by Percy, *op. cit.*, p. 48.

probability, increasing the volatilisation losses, since PbS is much more volatile than either the metal or its oxide.

At *Couëron* (France), until recently, rich galena ores from Sardinia were smelted in the modified Flintshire furnace (already figured), but the process was modified by lengthening its duration and lowering the temperature after the Silesian method, so that the charge was never allowed to melt. By these



Figs. 7 and 8.—Flintshire Furnace (Cordoba).

modifications in the furnace and in the process, the loss by volatilisation was reduced very considerably.

At the *Cordoba* Smelting Works (Spain) there are six furnaces of the modified Flintshire type shown in Figs. 7 and 8, which are built wholly of local (Seville) firebrick, costing about £7 per thousand delivered, upon a foundation of rough concrete, which is itself supported on a vault containing

water. This helps to cool the hearth, at the same time that it facilitates the discovery of any leaks. The actual working hearth is composed of blast-furnace black slag, which is smelted in upon a layer of well-rammed brasque, as shown in Fig. 7. The outside well or kettle has a separate fireplace.

Each of these furnaces works off three charges per 24 hours, each charge consisting of 1,600 kilos. (3,520 lbs.) galena averaging by iron pot assay  $77\frac{1}{2}$  per cent. lead, together with 70 kilos. (154 lbs.) fume. Each furnace is worked in eight-hour shifts by two men per shift, who are allowed about 60 kilos. (say 4 per cent. by weight) of lime, and 690 kilos. (say 43 per cent. by weight) of long-flame coal; they are paid ptas. 6.09 (say 4s. 4d.) per metric ton of lead produced, besides a bonus on the coal saved, which frequently amounts to as much as one-tenth of the total allowance, or even more, when the furnaces are new, so making the actual average consumption of coal over a long period from 37 to 40 per cent. by weight on the ore. The length of the campaign is variable, but averages six months.

The direct yield of lead averages 64 per cent. by weight, or 83 per cent. of the lead contents of the ore. In addition, 3 per cent. of the original lead contents are saved in fume, which amounts to 4.2 per cent. by weight of the original ore, and contains on an average 54 per cent. of lead, while another  $8\frac{1}{2}$  per cent. of the original lead contents is recovered from the grey slags, which amount to 20 to 30 per cent. of the weight of ore smelted, making the total recovery  $94\frac{1}{2}$  per cent. of the original contents.

The detail cost of smelting, together with sundry other details, will be found in Table V. The total cost given, however, amounting to 15s. 0 $\frac{1}{2}$ d. per ton, covers only the costs in the reverberatory furnace; and allows nothing for the handling of bye-products, smelting grey slags, cleaning flues, calcining matte, &c. If all these costs be included, together with administration charges, the cost is considerably over £1 per ton of ore smelted.

At *Desloge* (Missouri), according to Ingalls,\* the reverberatory process still survives in the shape of two furnaces of the Silesian form, 16 feet long by 11 feet wide, which are worked in the English fashion on eight-hour shifts. Each furnace is provided with a separate chimney, 55 feet high, and there are no flues for condensation of fume, so it may be seen that the process is wasteful. The grate is also exceedingly shallow, and there is much waste of fuel through the spaces between the bars. The working hearth is 8 inches thick of grey slag beaten down upon a brick basin, resting upon a layer of clay; the stoppages for repairs amount to about 20 to 25 days per year. Each eight-hour shift of two men works off a 3,500-lb. charge, but the ore being poor, the direct recovery is only 77 per cent. of the lead present. The grey slag is sold to blast-furnace lead smelters, who pay 90 per cent. of its lead value when it carries less than 40 per cent. of the metal. Each smelter gets \$1.65, and each fireman \$1.40 per shift. The ore is calcined without admixture of lime, and at the end of the roasting period from 85 to 100 lbs. of fluorspar (say from  $2\frac{1}{4}$  to 3 per cent. by weight) is scattered over the charge and thoroughly mixed with it by rabbling; the doors are then closed, and the whole of the charge run down into the well. The daily cost of smelting for five furnaces is given as follows:—

\* *E. and M. J.*, Dec. 5, 1905, p. 1111.

<i>Labour Cost—</i>		Dollars.	Dollars.
1 foreman at \$3.00,	.	3.00	
5 furnace crews at \$9.90 each,	.	49.50	
Unloading 21 tons coal at 6 cents,	.	1.26	
„ 14 tons lead at 15 cents,	.	2.10	
„ 7 tons grey slag at 15 cents,	.	1.05	
			56.91
<i>Materials, &amp;c.—</i>			
21 tons coal at \$2,	.	42.00	
Flux and supplies, iron, &c.,	.	13.00	
Blacksmithing and repairs,	.	10.00	
			65.00
			<u>\$121.91</u>

Per short ton \$4.65.

Other details of these furnaces and the work done in them will be found in Tables IV. and V.

*Spanish Furnaces.*—In the South of Spain a peculiar type of furnace, called the “boliche,” designed originally for burning brushwood, and built entirely of sandstone or other refractory stone, appears to have been developed locally from the original model of a baker’s oven. Such furnaces were formerly used by the *Alamillos* and *Fortuna* Companies at Linares, and are figured by Percy.\* In the old “boliche,” burning brushwood, ashes and fine cinders from the ashpit were exclusively used for the purpose of “drying” the charge and “setting up” for re-roasting. With the advent of English smelters and of coal as fuel, lime was substituted for ashes, and is now in general use, although fine ashes are still sometimes mixed with the lime.

Many modifications of the “boliche” are in use, approaching somewhat to the type of the Flintshire furnace. That shown in Figs. 9, 10, and 11† is from the *San Luis* Smelting Works at Linares. The outside of this is built entirely of the refractory sandstone which forms the common building stone of that district, the arch and lining being of firebrick, and the hearth of a brasque composed of refractory clay and fine coke breeze.

There is only one door, *a*, which serves for working as well as charging, and is situated immediately in front of the firebridge, *b*. The hearth, *c*, is of an unusual shape, and the flame enters it in a very peculiar way at one corner, curving round towards the flue, which for a short distance is double and then single, a wide flue-chamber at *d* being supposed to have an important effect upon the draught; so much so that different smelters have their own special fancies about the shape and size of the flue-chambers attached to their furnaces, which in some cases attain to exaggerated dimensions. The hearth is inclined steeply towards the interior well, *e*, in front of the single door, which is provided with counterbalanced iron sliding door, and the usual iron bar extends across to serve as a tool-rest. The cost of such a furnace erected in Linares is about £140.

**Mode of Working.**—The charge of the Spanish furnace figured is 3,372 lbs. of a pure galena with 78 to 80 per cent. of lead, and three charges are worked off in 24 hours by a smelter and an assistant per shift, with an extra assistant every 24 hours to make up the charge, clean up round the furnace, &c.

\* *Metallurgy of Lead*, p. 250.

† Sanchez y Massia, *Metallurgia del Plomo*, Madrid, 1893, p. 62.

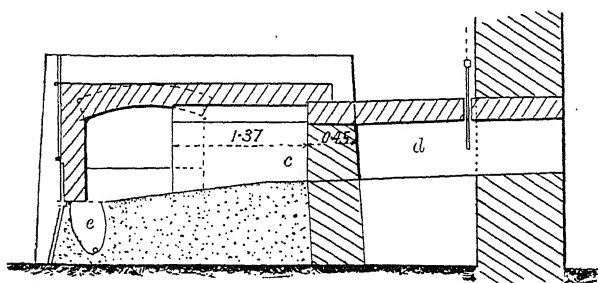


Fig. 9.—Section through AB.

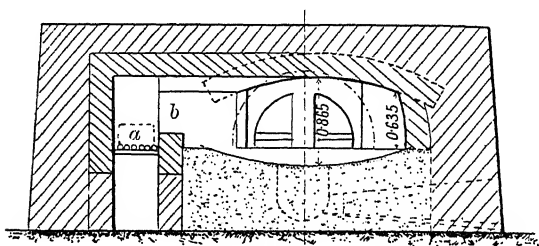


Fig. 10.—Section through CD.

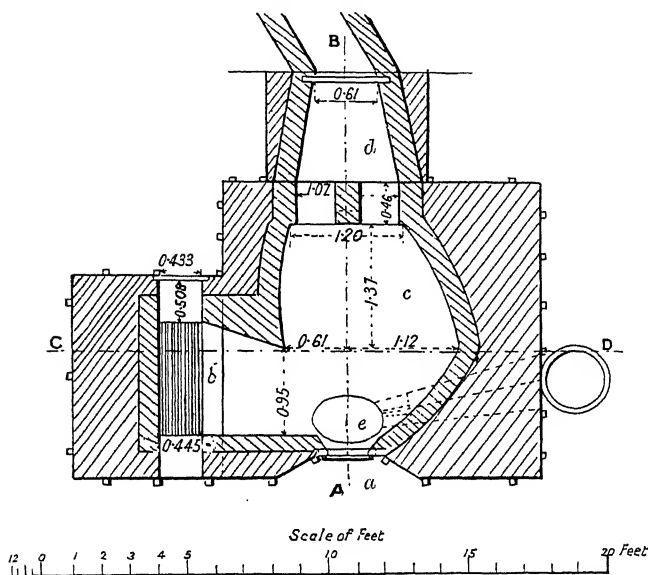


Fig. 11.

The charge is first spread over the hearth as evenly as possible, and the doors are kept open while the temperature is kept at a dull red. This first stage of *calcination* lasts from  $1\frac{1}{2}$  to 2 hours.



The firebars are cleaned, and the fire is made up, the smelter stirs the charge, and turns it over to expose fresh surfaces; after which the fire and furnace doors are closed, and the temperature is raised quickly to cause reaction and the *running down* of the charge to the interior well.

When this has taken place the doors are thrown open, and the slag or partially molten charge on top of the lead is "set up" and spread over the hearth, where it is re-calined with the furnace and fire doors fully open, this *second calcination* lasting about an hour. When the operation is judged to have reached a proper point the doors are again closed, and the fire urged to a *second running down* of the charge. The slags are then thickened with lime and withdrawn, the lead in the interior well is drossed, and it is then tapped into the outer well, from whence it is ladled into moulds. The whole operation in practice takes about 7 to 7½ hours.

The direct yield in lead on ores containing from 78 to 80 per cent. of that metal is about 62 per cent. of the weight of the ore, or, say, 80 per cent. of the total lead present. Of the remainder, 8 per cent. on an average is recovered from the slags, which contain on an average 50 to 55 per cent. of lead, 3 per cent. is collected in the fume, and the remaining 9 per cent. is lost. The fuel consumption is on an average 40 per cent., though with great care it may be cut down to 33 per cent.

The high volatilisation loss is due in part to the awkward position of the working door immediately in front of the firebridge, which also is accountable for the great amount of inconvenience and "leading" to which the workmen are exposed.

The direct cost of smelting in these furnaces for labour and fuel, lime and tools only, without any allowance for repairs or general expenses, is about 10s. per ton of ore treated.

**The Silesian Process** may be considered as the slow, low-temperature Carinthian process carried on with large charges in a modified English furnace, the final stage of slag reduction being omitted and carried on in cupolas. The use of a large furnace secures the principal advantages of the English process—namely, a large production at low cost for fuel and wages. The incomplete slag reduction is no drawback, since in any case reverberatory treatment of lead slags has always to be supplemented by a separate treatment in a hearth or cupola, by which means the volatilisation loss is lessened; and by stopping the reduction early no copper or antimony come down with the last lead to contaminate that already tapped. The lead produced in this process is, therefore, purer than in the English process. Moreover, the volatilisation loss of lead is less than by any other process, owing to the low temperature at which the roasting is conducted.

**The Silesian Process at Tarnowitz (Upper Silesia).**\*—The process at the Friedrichshütte, Tarnowitz, is a little peculiar in that a portion of the ore, specially rich in blende, is roasted as thoroughly as possible in a long reverberatory roaster and then added to the rest of the charge in the lead furnace, together with flue dust, litharge, and other oxidised products from the refinery, all of which assist in the reaction upon undecomposed lead sulphide.

The ores treated contain about 70 per cent. lead as galena, with 5 per cent. zinc as blende, and trifling admixtures of pyrites, chalybite, calcite,

\* *Zeitschrift für Berg-Hütten- und Salinen-Wesen*, vol. xxxii., p. 94, and vol. xxxiv., p. 292; also Schmabel, *Handbuch der Metallhüttenkunde*, vol. i., p. 305.

dolomite, and calamine. They are practically free from copper and arsenic, contain very little antimony, and only about 5 ozs. of silver per ton.

The construction of the furnace used is shown in the following Figs. 12

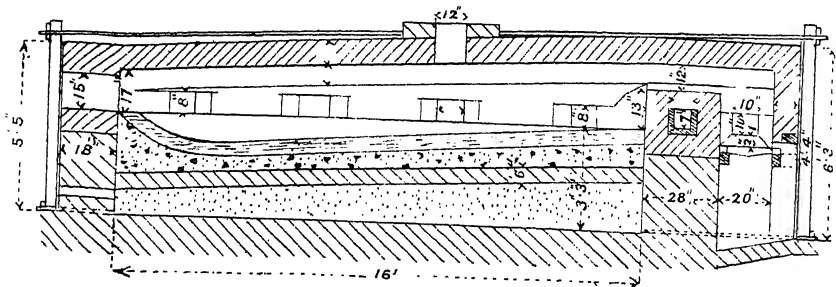


Fig. 12.—Silesian Furnace (Longitudinal Section).

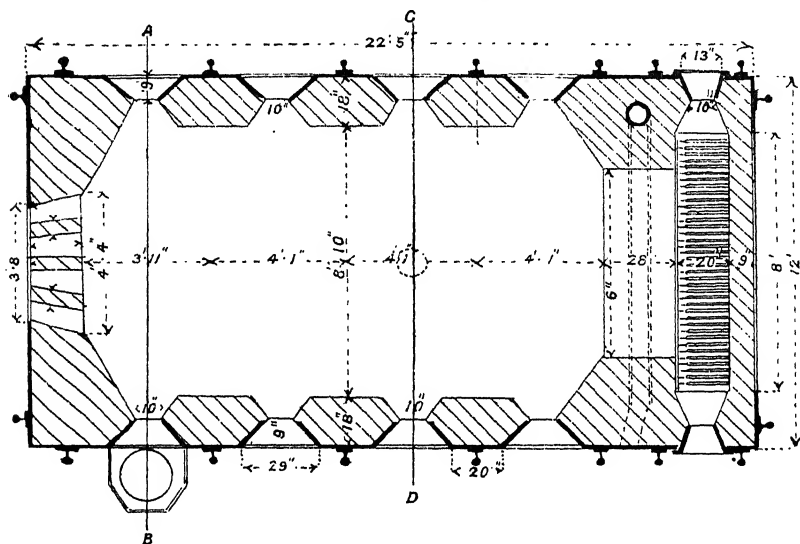
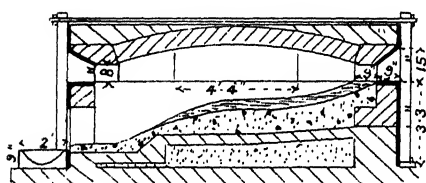
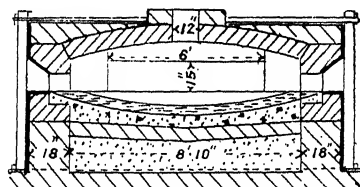


Fig. 13.—Silesian Furnace (Plan).



Section A. B.

Fig. 14.—Silesian Furnace.



Section C.D.

Fig. 15.—Silesian Furnace.

to 15. In general form it somewhat resembles the English furnaces already described, especially the Couëron modification of the Flintshire furnace, for the hearth slopes gradually towards the sump, and not suddenly as in the

old Flintshire furnace, and the well is at the farther end door. It is rectangular in form, with four working doors on each side, the well being at the coolest end of the furnace below the door next the flue, the opening into which is divided into four by means of brick partitions. The bridge is hollow, and contains a cast-iron trough, cooled by a current of air which is drawn through it by means of a hole in the roof, as shown on the plan, Fig. 13. The iron-grate bars are placed transversely in the firebox, and not longitudinally as usual, firing takes place from both sides, ashes being removed at the end of the furnace. The roof is horizontal from the firebox to the centre of the hearth, where the charging opening is situated, falling thence in a straight line to the flue (longitudinal section, Fig. 12). The furnace is completely cased in iron plates strongly bound by iron-rail buckstays and tie-rods. The hearth is built of red brick on a thick sand bed sloping towards the flue at about the right angle for the final hearth, as shown in Fig. 14. Over this is formed the trough-like brasque bottom, composed of clay 1 part, coke 3 parts, which slopes towards the well, and varies in thickness from 4 inches to 12 inches. The actual working bottom is a layer 4 inches thick of slag having a high melting point, and spread evenly over the whole area, following the surface of the brasque bottom already rammed to the proper curve. This working hearth used to be made of tap cinder, but ordinary gray slag mixed with lime is now used instead, exactly as in English furnaces.

**Mode of Working.**—The charge weighing 2,500 kilos. (say  $2\frac{1}{2}$  tons English) and crushed to pass a 5-mesh screen, is dropped into the furnace, which is already at a dark red heat ( $500^{\circ}$  to  $600^{\circ}$  C.), and spread out so as to form a layer 3 inches to 4 inches thick. The fire is fed with cinders only, so as to keep the temperature low; the galena begins to decrepitate, and the charge gradually acquires the temperature of the furnace after half an hour. The doors are now opened, and roasting begins at the surface of the charge, a white crust of oxide and sulphate being formed. About every twenty minutes the charge is turned over by means of paddles, or stirred with rabbles, so as to expose new surfaces to the action of the air and prevent clotting. After four hours the oxidised additions are made, consisting of a portion of the same ore previously roasted, with flue dust and carbonate ore when available. The total weight of these additions is about 10 to 12 cwt., and they are well stirred in; at the same time the grates are cleaned and fed with good coal, and the damper is fully opened so as to raise the temperature. The reaction period begins, lead runs down, and the ore softens so much as the temperature rises that lime has to be thrown on the charge and well rabbled in to stiffen it, after which the whole charge is turned over. The working doors being open during these operations, the furnace cools off so much that the flow of lead ceases, while more of the undecomposed sulphide oxidises. On shutting the furnace doors the temperature soon rises, reaction begins again, and more lead flows out. This alternate oxidation and heating of the charge takes place at intervals of about one and a-half to two hours throughout the whole period of reaction. About three hours from the start the well is tapped into the kettle, poled there, drossed, and ladled into moulds, the dross being set aside for subsequent treatment. This first kettleful of "virgin lead" carries from 50 to 60 ozs. of silver per ton. About 10 cwt. of oxidised skimmings from the steam refining of lead desilverised by the Parkes process, and containing 75 to 80 per cent.  $PbO$  and 18 to 20 per cent.  $ZnO$ , are now

added and the doors closed, while the fire is fed with small coal to ensure a smoky flame. This reduces lead sulphate to sulphide, and lead oxide to metal, and the flow of lead begins again. Further tapplings are obtained each hour and a-half, steadily decreasing, however, in amount and in silver contents. Before the last tapping the kettle drosses obtained by poling the previous kettle leads are thrown into the furnace and melted down with the last tapping. In all, the reaction period lasts seven hours; after which the residues are raked out, the hearth repaired where necessary and sprinkled with lime, and a new charge added.

The removal of the residues was formerly effected by raking them out of the last working door opposite the well into a box filled with water, but the fumes were very injurious to the workmen employed, and dangerous explosions sometimes occurred. In the latest furnace, the residues are raked through an opening in the furnace hearth, which is covered by an iron plate during the working of the charge, into a wheeled iron slag pot underneath. This improvement, not shown in the figure, has had a great effect on the health of the workmen.

About 60 to 65 per cent. only of the lead contents of the ores are extracted as work-lead, containing on the average 6 to 7 ozs. silver per ton; the remainder is all found in the residues (amounting to 30 to 35 per cent. of the weight of the charge and containing 40 to 55 per cent. metal), except about 4.45 per cent. Of this, most is recovered in the form of flue-dust, which chiefly consists of  $\text{PbSO}_4$ , so that the real loss in the process unaccounted for is only  $1\frac{1}{2}$  per cent.\*—calculated on the commercial dry assay of the ore. The residues are treated in a cupola furnace.

The total length of an operation is 11 to 12 hours, two men working on each 12-hour shift. The consumption of fuel varies between 40 and 50 per cent. of the weight of the charge, and that of lime amounts to about 1 per cent.

At *Bleiberg near Montzen* (Belgium)† the furnace in use is similar, except that it has a fireplace at each end, the flue opening being in the middle of the arch. The calcination period on a 2-ton charge lasts six hours, and the period of reactions six hours. Further details are given in Tables IV. and V.

**The Flowing or Cornish Process.** In this process, now extinct, the operations of calcination and of reaction were performed in separate furnaces, and a larger quantity of  $\text{PbO}$  and  $\text{PbSO}_4$  was formed in the first than was required for reaction upon  $\text{PbS}$  in the second. The hearth of the calcining furnace was rhomb-shaped, the firebridge and flue door being at the ends of the longer axis, and the working doors at the ends of the transverse axis. The charge was from  $1\frac{1}{2}$  to 3 tons, according to size, and the calcination took 15 to 18 hours with a coal consumption of 30 per cent. by weight.

The flowing furnace is rectangular and longer, and contains four working doors on opposite sides, besides the flue door at the end opposite the firebridge. The hearth is saucer-shaped in longitudinal section, and wholly inclined sideways towards the second door on the front, beneath which is the usual cast-iron pot. The whole of the hearth is carried on iron bars

\* Schnabel, *Handbuch der Metallhüttenkunde*, vol. i., p. 307.

† Schnabel, *op. cit.*, p. 308.

stretched transversely across the foundation walls and arranged roughly parallel to the contour of the working bottom.

The charge in the Cornish process was 2 tons of calcined ore, which was spread on the hearth and run down into the well in about three hours, when the lead was tapped out, the residues thickened with lime and culm and "set up." Some 5 per cent. of scrap iron was then thrown into the well, and the charge again melted down, the lead being tapped out together with the matte, or "slurry," formed, which carried nearly all the copper present in the ores. The whole time required to work off a charge was about eight hours, and the coal consumption 40 per cent. The slags were usually poor enough to throw away, and the process was specially adapted to the smelting of coppery ores containing only 60 to 70 per cent. lead.

The whole operation in the flowing furnace usually took about eight hours, and on account of the higher temperature required to melt the larger quantity of refractory material, the volatilisation loss and consumption of fuel were both greater than in the Flintshire furnace, which more than offset the advantage of cleaner slags. The flowing furnace has been, however, very largely used in Wales and elsewhere for reducing the grey slags of the Flintshire furnace, as well as for treating rich silver ores, lead residues, oxidised refinery products, jeweller's "sweep," &c., and is still in use to a considerable extent for such and similar purposes.

TABLE IV.—DETAILS OF REVERBERATORY LEAD FURNACES.

	Cornithian Method.		English Method.				Silesian Method.	
	Rail, Carinthia.	Granby, Missouri.	Holywell, Flintshire.	Cardoba, Spain.	Tarnowitz, Silesia.	Desloge, Missouri.	Tarnowitz, Silesia.	Montzen, Belgium.
Hearth length, . . .	11' 0"	10' 0"	12' 0"	10' 6"	11' 9"	16' 0"	16' 0"	15' 5"
„ width at bridge, . . .	} 5' 0" {	2' 6"	3' 6"	4' 11"	} 10' 10" {	11' 0"	8' 10"	9' 6"
„ „ flue, . . .		4' 0"	4' 0"	3' 11"				
„ medium width, . . .		2' 0"	9' 6"	8' 10"				
Grate length, . . .	7' 0"	3' 0"	4' 6"	7' 0"	8' 0"	8' 0"	8' 0"	6' 6"
„ width, . . .	1' 3"	2' 0"	2' 6"	1' 10"	1' 8"	3' 0"	1' 8"	2' 8"
Bridge length, . . .	3' 7"	2' 0"	3' 6"	4' 11"	6' 0"	...	6' 0"	6' 0"
„ height above hearth, . . .	4'-5"	...	12"	12"	1' 3"	...	1' 3"	...
„ height to roof, . . .	9"	6"	9"	1' 10"	1' 0"	...	1' 0"	1' 0"
Grate area : hearth area, . . .	1 : 6	1 : 5.4	1 : 10	1 : 9.4	1 : 9.6	1 : 7.5	1 : 10.6	1 : 8.3
„ chimney section, . . .	7 : 1	8 : 1	1.1 : 1	...	...	...	...	...
Hearth area (sq. ft.), . . .	55	31	94.5	100	127	170	140	141

**Comparison of the Various Processes.**—Comparative data have been thrown into tabular form to facilitate reference. Table IV. gives constructional details of various lead furnaces, and Table V. details of the operations carried out in them.

TABLE V.—COMPARATIVE DATA OF REVERBERATORY LEAD SMELTING.

	Carinthian Method.			Flintshire Method.				Bolche.	Silesian Method.	
	Raibl, (artificial).	Granby, Missouri.	Holywell, Flintshire.	Conétron, Loire Infer.	Tarnowitz old furnace.	Cordoba, Spain.	Desloge, Missouri.		Tarnowitz new furnace.	Montzen, Belgium.
Reference,	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.
Charge—weight in lbs.,	370-430	1,500	2,352	2,970	4,400	3,526	3,500	3,372	6,820	4,580
lbs. per square foot of hearth,	7-8	48	25	...	35	35	21	...	49	32-5
" assayed lead,	70	...	77½	81	73	77-5	70	78-80	70	80
" silver,	...	...	...	...	24-4	10	...	...	...	...
Duration of operation,	10-12	9-11	5½	9½	7	7½	8	...	12	16
hours,	1½	2	6	...	6	6	...	7½	4	...
Men per furnace,	230-270	1,575	1,575	...	2,807	2,257	1,900	2,130	...	2,308
Lead produced per charge,	...	...	308	638	700	750-950	945	...	1,700-2,300	1,456
lbs.,	95	...	13-1	21-5	16	21-28	27	...	23-35	33
Grey slag produced per charge,	24	...	55-5	50	38-8	26-33	38	...	45-46	66
per cent. of charge by weight,	4	...	...	...	121	164	...	...	...	...
" assayed lead,	...	...	...	...	27½	44	...	...	...	...
Flue dust per charge,	...	...	...	...	50	54	...	...	61	...
lbs.	...	...	...	...	...	...	...	...	...	...
per cent. of charge by weight,	...	...	...	...	...	...	...	...	...	...
" assayed lead,	...	...	...	...	...	...	...	...	...	...
Of lead charged :—	...	...	...	...	...	...	...	...	...	...
Obtained in pigs,	93	...	87-0	82-4	87-4	83-0	77-7	80	60-65	66-1
from grey slag,	...	...	7-0	12-4	8-0	8-5	14-8	8	30-35	27-8
per cent.,	...	...	...	...	...	...	...	...	...	...
" from flue dust,	...	...	1-0	5-2	1-4	3-0	...	3	...	...
per cent.,	...	...	...	...	...	...	...	...	...	...
Volatilisation—loss,	7*	...	5-0	...	3-2	5-5	7-5	9	13-2	5-1
per cent.,	...	...	...	...	...	...	...	...	...	...
Fuel consumption—coal,	...	...	50-0	40-0	...	37-40	80-90	40	40-50	40-0
per cent.,	...	...	...	2-0	1-0	4-0	...	...	1	...
Consumption of lime,	...	...	s. d.	...	s. d.	s. d.	s. d.	s. d.	...	...
Cost of smelting—per ton :—	...	...	...	...	...	...	...	...	...	...
Superintendence, &c.,	...	...	...	...	...	0	84	...	...	...
Labour,	...	...	...	...	...	2	11½	...	...	...
Coal,	...	...	...	...	...	6	11½	...	...	...
Lime, and other flux if any,	...	...	...	...	...	0	8	...	...	...
Tools, repairs, and general expenses,	...	...	...	...	...	0	4½	...	...	...
	...	...	...	...	...	4	11½	(?)	...	...
Total,	...	...	9:6	...	9:0	15 0½	22 3	10:0	...	...

\* Total losses.

References.—1. Hofman, *Met. of Lead*, 1893, p. 107. 3. Percy, *Met. of Lead*, 1870, p. 231. 4. Phillips, *Elements of Metallurgy*, 1879, p. 540. 5. Percy, *op. cit.*, p. 243, and Hofman, *op. cit.*, p. 107. 6. *Private Notes*, 1908. 7. Ingalls, *Eng. and Min. Journ.*, Dec. 1905, p. 1111. 8. Sanchez y Massia, *Metadurgia del Plomo*, 1893, p. 62. 9. Schnabel, *Handbuch der Metallhüttenkunde*, 1894, p. 305. 10. *Ibid.*, p. 308.

## Bye-Products and their Treatment.

The principal bye-products of reverberatory lead smelting are grey slag and fume.

*Grey Slag.*—The composition of this substance varies considerably, as shown by the following analyses :—

TABLE VI—ANALYSES OF GREY SLAGS. \*

	Carinthian Furnace, Engin.	Air Furnace, Missouri.	Flintshire, Holywell.	Flintshire, Bagillt.	Silesian, Tarnowitz.
Reference.	1.	2.	3.	4.	5.
Pb (free), . . .	...	...	...	...	...
PbS, . . .	...	20·929	0·90	...	...
PbSO <sub>4</sub> , . . .	...	2·349	9·85	18·73	13·269
PbSiO <sub>3</sub> , . . .	45·05	44·261	59·28	48·28	12·373
PbO, . . .	...	...	2·11	...	24·375
ZnO, . . .	20·80	7·146	7·52	12·00	22·857
Al <sub>2</sub> O <sub>3</sub> , . . .	14·70	{ 0·152	3·01	0·71	tr.
Fe <sub>2</sub> O <sub>3</sub> , . . .			2·86	...	...
FeO, . . .	...	...	...	3·89	8·957
FeS, . . .	...	...	...	...	1·823
MnO, . . .	...	...	...	...	tr.
CaO, . . .	3·70	4·650	12·68	13·50	11·190
MgO, . . .	...	3·948	...	...	...
C, . . .	...	...	...	...	4·821
Ag, . . .	...	...	...	...	0·015
S, . . .	0·10	...	...	...	...
SiO <sub>2</sub> , . . .	15·85	12·049	...	2·12	...
Insoluble, . . .	...	...	1·45	...	...
Total Pb, . . .	99·80	99·063	99·66	99·23	99·680
	...	...	52·88	...	...

*References.*—1. Bouhy, *La fabrication du plomb*, 1870. 2. Williams, *Geol. Survey of Missouri*, 1877, pp. 8-101. 3. Percy, *Metallurgy of Lead*, 1870, p. 235. 4. *Ibid.*, p. 239. 5. *Ibid.*, p. 242.

It is sometimes treated in the flowing furnace, sometimes in the slag hearth (a form of low, open, blast furnace), more commonly, however, in some form of cupola or high blast furnace.

*Fume.*—Some analyses of this substance from reverberatory furnaces are given in Chapter xii., and the various methods of collection and condensation are also dealt with in that chapter. Various modes of “bricking” it, as a preparation for further treatment, are also referred to in the same chapter. At a large number of reverberatory smelting works the fume is bricked and run down in a cupola together with the grey slag, but, owing to its light character, it is best treated in reverberatories mixed with the ore, since it serves to shorten the roasting period and facilitates the reactions.

At the *Cordoba Smelting Works*† grey slags from the modified Flint-

\* *v.* Table IX. for an analysis of imperfectly fused grey slag from hearth smelting.

† *Private Notes*, 1908.

shire furnaces described and figured are smelted in a slag-cupola similar in general to that shown in Figs. 75 and 76, built of refractory grit cut to shape by contract, 15 or 16 stones going to form the circle 7 feet 6 inches in outside diameter, each ring being from 5 to 6 inches high, and varying in thickness from 2 feet near the bottom to 18 inches at the top. There are two of these cupolas, one with five and the other with six water-tuyeres, one being 11 and the other 45 inches in diameter at tuyere level, while the height is in each case about 6 feet from tuyeres to top of charge-column at the feed-door. The charge is made up together with the fuel into beds, a typical bed consists of grey slag  $38\frac{1}{2}$  tons, iron ore  $15\frac{1}{2}$  tons, limestone  $8\frac{1}{2}$  tons, foul slag (from previous operations)  $10\frac{1}{2}$  tons, and coke  $10\frac{1}{2}$  tons. The fuel consumption is thus about  $14\frac{1}{2}$  per cent. by weight on the charge. Blast is furnished to the two cupolas by a Thwaites' No. 4 Roots' blower, running at 100 to 120 revolutions per minute. Lead is tapped once or twice a shift into a pot sunk in the ground by the side of the furnace, whence, after drossing, it is cast into moulds. A syphon-tap (*v. Chap. viii.*) was tried on these cupolas, but did not work well, because the men dipped out too much lead, and let down the charge, with the result that the tap froze and gave trouble, so tapping from the bottom of the crucible was again resorted to. Slag runs almost continuously from a taphole into long conical iron pots of the old German pattern, which are removed at intervals by means of an independent carriage, and allowed to cool. Near the point of the cones a good deal of matte accumulates. This is roasted in heaps and returned to the furnaces, together with the dross, &c. Each furnace puts through 35 to 38 tons of charge per day, and requires six men per eight-hour shift, besides a foreman, making the cost for labour  $\frac{2}{3}$  of a man per ton smelted, or at the prices paid, say  $10\frac{1}{4}$ d. per ton. The better direct yield of silver in the original smelting as compared with that of lead is exemplified by the fact that, while the ore lead from the reverberatories contains 12 to 14 ozs. silver, that from the slag cupolas contains only  $3\frac{1}{2}$  to  $4\frac{1}{2}$  ozs.

The cost of smelting per ton of grey slag smelted at the prevailing prices for fuel and fluxes, not including the cost of blast, is as follows :-

			Per Ton Grey Slag.	
			Pesetas.	Sterling at 2s.
Labour (including repairs),	.	.	4.82	£0 3 5
Iron flux, at 19.60 pesetas = $14\frac{1}{2}$ per ton,	.	.	7.91	0 5 8
Limestone, at 3.00 „ = $2\frac{1}{2}$ „	.	.	0.65	0 0 6
Coke, at 55.20 „ = $39\frac{5}{8}$ „	.	.	14.80	0 10 7
			<u>28.18</u>	<u>£1 0 2</u>

It is clear that some economy could be effected on these figures by somewhat increasing the proportion of limestone and decreasing that of iron flux, but it would be necessary to employ a water-jacketed furnace. The height of the charge-column could also be increased with great advantage as regards fuel consumption and volatilisation loss.

The "black slag" produced from such slag-cupolas as those described is of variable composition, depending not only upon the nature of the gangue associated with the original ore, but upon the composition of the fluxes employed, and even that of the material of which the furnaces were built.



The following Table VII. gives the composition of a few samples of black slag from slag-cupolas :—

TABLE VII.—ANALYSES OF BLACK SLAGS.

	Cordoba, Spain.	La Tortilla, Linares, Spain.	Petraeus, Joplin, Mo.
Reference.	1	2	3
SiO <sub>2</sub> , . . . . .	33·75	39·80	22 to 28
FeO, . . . . .	19·45	12·10	31
MnO, . . . . .	1·78	0·50	...
CaO, . . . . .	19·75	30·90	20 to 24
MgO, . . . . .	2·82	...	...
Al <sub>2</sub> O <sub>3</sub> , . . . . .	17·55	12·97	...
ZnO, . . . . .	...	...	7
S, . . . . .	0·50	...	...
Pb, . . . . .	0·30	3·61	0·7
	95·90	99·88	

References.—1 and 2. Author's Laboratory, 1908. 3. *E. and M. J.*, November 9, 1907, p. 867.

## CHAPTER IV.

### LEAD SMELTING IN HEARTHS.

THE "hearth" is a species of low furnace, the best known example of which is the blacksmith's forge, in which the charge, mixed with fuel or in actual contact with it, is handled by appropriate tools, while the heat is kept up by the action of a blast on the fuel. It may be said, therefore, to be in its operation intermediate between a reverberatory and a blast furnace. The mixture of fuel and ore floats on a bath of molten lead, of which the hearth is kept full so as to avoid the formation of crusts. The "roast and reaction method," as carried on in these furnaces, differs essentially from that in reverberatories, because roasting and reaction go on simultaneously; and there is considerable direct reduction of lead oxide (formed by the action of the blast on galena) to metal by the glowing fuel. As regards purity of the ore adapted to the process, the same conditions which govern its applicability to reverberatory treatment must be even more strictly observed; while, the volatilisation loss being much higher, argentiferous galenas are always better treated in reverberatories. Finely-powdered ore, moreover, if treated in a hearth without previous agglomeration or partial roasting, gives rise to considerable loss, as it is sent into the flues by the blast. Ores containing 80 per cent. of lead will yield "direct" in the hearth from 56 to 63 per cent., the remaining 17 to 20 per cent. going to slag and fume; whereas ores containing 65 to 70 per cent. lead will not yield more than 42 to 44 per cent. direct, the amount passing into the slag and fume being no less than one-third of the total lead contents of the ore.

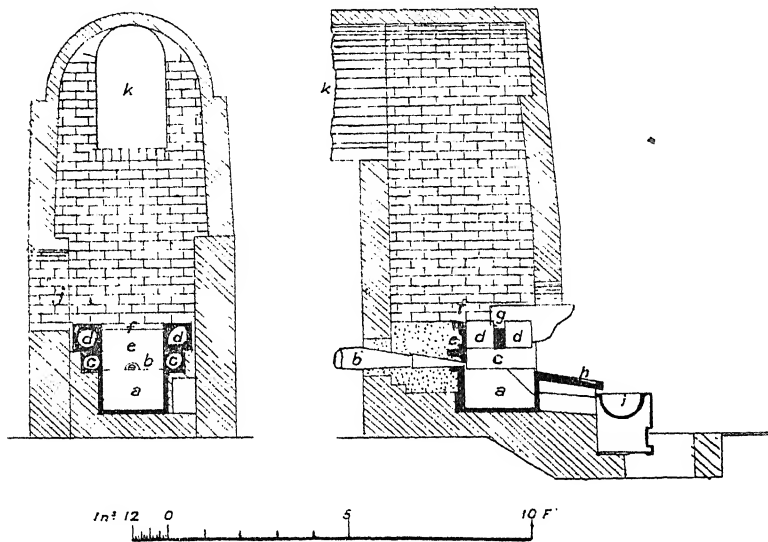
The capacity of the hearth is less than that of the reverberatory, and it requires power and a blower. On the other hand, the fuel consumption is less (usually not more than one-half) any kind of fuel can be used, from coal to wood or peat; and it possesses the great advantage over a reverberatory of being easily stopped and started again without much loss of time or of heat, and without those strains caused by alternate expansion and contraction which are so destructive to the brickwork of a reverberatory. Very perfect arrangements are necessary for drawing off and condensing as much as possible of the volatilised lead fume, otherwise the operation of the hearths is dangerous to the men employed, who are more exposed to the fumes than are those working reverberatory furnaces.

The hearth is invariably made of cast iron, and in front of it is a cast-iron "workplate," with a groove to carry off the reduced lead into a kettle. There is usually a partial front wall composed of a thick bar of iron, but often the front is simply a dam formed of ashes and powdered galena which is impervious to melted lead. The sides and back must always be of iron, in order to admit of chipping off the half-fused masses of slag which constantly adhere to them; they may be cooled simply by radiation from large and

heavy castings (Scotch hearth), by a current of air (Rossie hearth), by a water-box (American hearth), or by both air- and water-boxes combined (Moffet-Jumbo hearth). These differences may conveniently be taken as representing four distinct modifications of the hearth process, though the manipulation is much the same in all.

1. **The Scotch Hearth.**—This furnace was invented many centuries ago in the Leadhills (Scotland), where it is still in use, the principal improvements being the substitution of heavy castings for the flagstones and thin castings originally used to form the sides and back of the furnace, by which means the working of the furnace, formerly intermittent with periods of "cooling off," in order to protect the furnace from destruction, can now be carried on continuously from Monday till Saturday. The fuel employed was formerly peat, but coal is now almost exclusively used.

*Construction.*—The construction of an ore hearth will be understood from the following Figs. 16 to 20. Figs. 16 and 17\* represent the ore hearth of



Figs. 16 and 17.—Keld Head Ore Hearth.

the Keld Head Co., Yorkshire, and Figs. 18 to 20† that in use at Leadhills, which may, perhaps, be regarded as the typical form of Scotch hearth. The lettering in both series of figures is the same to facilitate reference. The base is formed of a cast-iron "hearth bottom" or "hearth box," *a*, cast in one piece; its dimensions are, at Keld Head, 23 inches long, 21 inches wide, and 12 inches deep, the iron being  $1\frac{1}{2}$  inches thick in front and  $2\frac{1}{2}$  inches at the back; at Leadhills the box is 22 inches long from front to back, 30 inches wide, and only 6 inches deep, the iron being 1 inch thick throughout. This hearth bottom forms the crucible of the furnace, and, when running, is always full of molten lead. The dimensions of the hearth in use at Weardale are

\* Percy, *Metallurgy of Lead*, p. 281.

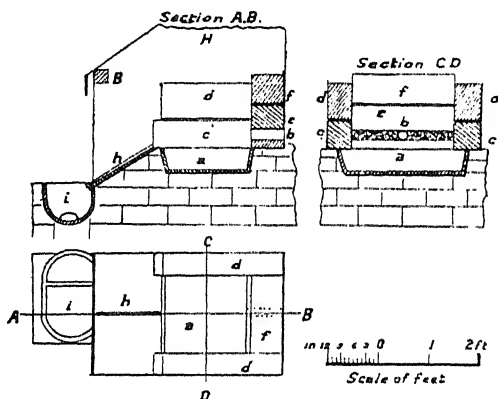
† Sexton, *E. and M. J.*, Feb. 23, 1895.

intermediate, being length front to back 21 inches, width 27 inches, depth 8 to 9 inches, capacity about  $\frac{3}{4}$  ton of molten lead. The hearth proper, in which the smelting takes place, is built upon it by means of heavy blocks of cast-iron, technically called "stones." In front of the hearth is the "workstone," *h*, which may be cast in one piece with the hearth bottom, as at Keld Head, but is sometimes a separate casting, and is provided with a ledge and a gutter to convey the molten lead to the melting pot in front. At the back is a single tuyere, *b*, which is bedded in a mixture of mortar and slag. The remaining pieces forming the hearth are as follows:—

*cc.* Two "bearers" or "lower sidestones" resting on the hearth box and measuring, at Keld Head, 2 feet long by 7 inches square (at Leadhills, 2 feet 9 $\frac{1}{2}$  inches long  $\times$  6 inches square).

*dd.* Four keystones 10 inches cube (Keld Head), or two "upper sidestones" 2 feet 6 inches long  $\times$  6 inches wide  $\times$  10 inches high (Leadhills).

*e.* "Lower backstone" or "pipestone," 2 feet 6 inches  $\times$  1 foot 2 inches  $\times$  3 inches thick at Keld Head; and 2 feet long  $\times$  8 inches square at Leadhills.



Figs. 18 to 20.—Scotch Hearth as used at Leadhills.

*f.* "Upper backstone," 2 feet 8 inches long  $\times$  5 inches wide  $\times$  2 inches deep at Keld Head; 2 feet long  $\times$  8 inches square at Leadhills.

*g.* "Forestone," which at Keld Head is a block fitting between the two keystones and at Leadhills is absent.

In the figures the lead pot, provided with small fireplace, is shown at *i*, that at Leadhills having a transverse partition perforated at the bottom to allow scum, ashes, &c., to float on the surface of the lead in the larger division, while only clean lead flows under into the smaller division, from which it is ladled at intervals. The hearths at Keld Head are arranged under brick hoods as shown, *j* being a working door at the side through which fuel is charged, and *k* the connection with the main flue. At Leadhills a sheet-iron hood, *H*, supported on an iron crossbar, *B*, and a brick back wall, carries off the fumes.

At *Weardale* (Durham)\* upon the hearth bottom, 21  $\times$  27  $\times$  8 inches,

\* *E. and M. J.*, July 6, 1905, p. 11.

are built two side stones, 27 to 28 inches long, 9 inches thick, and 15 inches high, cast hollow, and cooled by means of air which is caused to circulate through them. In place of the simple round or oval pipe tuyere (often  $2\frac{1}{2}$  inches high by 4 inches wide) of the Leadhills and other original Scotch hearths, which passes between the pipestone and the hearth, the Weardale tuyere enters the pipestone through a semi-circular opening on the back, which gradually widens out in its passage through the stone to a slit 12 inches long by 1 inch high, through which the blast emerges into the hearth, the slit itself being protected by a projecting ledge  $1\frac{1}{2}$  inches wide immediately above it. The pipestone is cast hollow, and the water supply for the water-box, into which the red-hot slag is thrown by the smelters, passes through it in order to cool it.

*Mode of Working.*—Each furnace requires two men on a shift, the shifts being formerly of eight hours, but now very commonly of six hours, or in some places even of four hours only; the tools used are paddles, shovels, and rakes; piles of ore, coal, and lime are arranged close in front of the hearth. After lighting a wood fire in the sump, coal is added under a light blast, continually turned over, and the ashes removed till the hearth is full of glowing fuel. If the hearth was not left full of lead, pigs are melted down until there is a sufficient bed below the fuel. A charge of about 20 lbs. of previously damped ore is then scattered on the bed of fuel, where it soon commences to yield lead and to slag. The bed of fuel is then lifted and stirred, the half-melted masses called “browse” are drawn out on to the workstone and broken up with the bar, completely slagged portions being thrown aside, while partially decomposed portions are returned together with more ore, and this process is continued until the hearth becomes full of melted lead, upon which float the remaining fuel and half-decomposed residues. The hearth, being now in normal working order, receives new charges at intervals of about five minutes. Each charge consists of a shovelful of coal, which is sprinkled over the residues, and thrown chiefly towards the back of the hearth near the tuyere, followed by from 20 to 30 lbs. of ore previously mixed with from  $1\frac{1}{2}$  to 2 per cent. of lime, and thoroughly damped. After adding each charge the fire is left undisturbed for about three minutes, one of the furnacemen then inserts his bar into the lead and raises the crusts lying beneath the fuel, while his companion with a long-handled shovel draws them out upon the workstone. The half-fused masses are broken up, slag is thrown into a water box at one side, while the half-decomposed ore is returned to the furnace with a little lime. The tuyere is cleaned out if necessary, a shovelful of coal added, and followed by another charge of ore.

The ores treated are pure galenas dressed by jig and buddle up to a minimum of about 77 per cent. lead; and the lead is very pure, seldom requiring to be poled before ladling, though this is sometimes done to improve the quality. Each hearth treats  $6\frac{1}{2}$  to 7 tons of ore per 24 hours, yielding about 4 tons (80 pigs) of lead, and the consumption of fuel is about 18 cwt. to 1 ton of coal slack, or say 10 to 12 per cent. of the weight of ore treated.

The grey slag produced is broken up and hand-picked to remove pieces of lead and of half-decomposed ore, but still retains many small shots of metal. After separating these, a sample of Leadhills slag yielded on analysis\* PbS 5·63 per cent., PbSO<sub>4</sub> 10·36 per cent., PbO 34·88 per cent. (equal

\* *E. and M. J.*, Feb. 23, 1895.

to total combined lead 34·8 per cent.), CaO 4·10 per cent., ZnO 0·95 per cent., Fe<sub>3</sub>O<sub>3</sub> and Al<sub>2</sub>O<sub>3</sub> 18·20 per cent., SiO<sub>2</sub> 26·00=100·12. It is allowed to accumulate till a sufficient quantity is obtained for a campaign, and is then run through a slag hearth. The fume, drawn off through the hood by means of the draught caused by a high chimney, is condensed—usually in chambers filled with brushwood through which a spray of water is continually falling. The water is led into a system of tanks, where it deposits a fine grey sludge which is plastered on the work plate to dry, and is then mixed with the ore charge in small proportions.

*Examples and Cost of Working.*—At some works in the North of England the ore is all smelted raw, at other works treating apparently similar ores they are roasted for about two hours prior to treatment. According to Middleton\* the following are the comparative results shown at two different works by treating 80 per cent. galenas in furnaces of the same dimensions with and without previous roasting respectively.

TABLE VIII.—COST OF SMELTING IN ORE-HEARTH.

	Ore, Raw.		Ore, Roasted.	
Weight of ore smelted per 24 hours, . . .	7½ tons.		9½ tons.	
Weight of lead produced per 24 hours, . . .	90 cwts.		120 cwts.	
Direct yield of lead on ore, . . . . .	56%		65%	
Proportion of lead sent into fume, . . . .	9%		6½%	
„ „ „ slag, . . . . .	15%		8½%	
Costs per ton of lead produced:—	lbs.	s. d.	lbs.	s. d.
Roasters' wages, . . . . .	...	...	...	2 6
Coal for roasting, . . . . .	...	...	425	2 8
Smelters' wages, . . . . .	...	8 5	...	4 5
Coal for smelting, . . . . .	250	1 7	75	0 5½
Peat and lime for ditto, . . . . .	.	0 2	...	0 4
Total cost, . . . . .	...	10 2	...	10 4½

At *La Tortilla Works*† (Linares, Spain) there are 24 plain Scotch hearths, 18 of which are in two rows back to back. The hearths are not water-cooled, and the bottom of each is cast in one piece with the front plate.

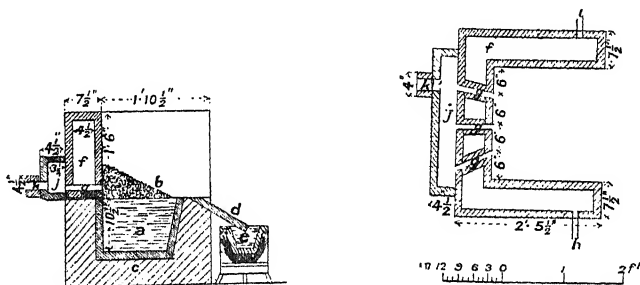
Each hearth is worked by five shifts of two men, each working four hours, so that the total working time of each hearth is 20 hours, with four hours free for weighing up, &c.; and on each shift the task is to smelt 1 metric ton of ore (to which is added 2 per cent. of slaked lime), besides cleaning off the slag from the tump stone and digging out any half-fused masses from around the hearth. Besides 2 per cent. of slaked lime, 200 kilos. of fume is well mixed with each 1,000 kilos. ore, which facilitates the reactions and saves roasting. Bituminous coal is used for fuel, and the consumption is well under 7 per cent. by weight. Smelting is all done by piecework, the price being ptas 17·00 (say 12s. 2d.) per ton of lead produced, which is separately weighed in from each hearth every 24 hours. The direct yield in the hearth

\* *E. and M. J.*, July 6, 1905, p. 11.† *Private Notes*, 1907.

is 62 to 63 per cent. on the weight of ore; the production of fume is considerable, but as this is returned continually to the operation, it is not taken into account, and the final volatilisation loss is only from 1 to 2 per cent. on the dry assay, say from  $2\frac{1}{2}$  to  $3\frac{1}{2}$  per cent. on the true contents of the ore.

2. **The Air-cooled or Rossie Hearth.**—In this form of hearth, first introduced at Rossie, N.Y., and afterwards in the Mississippi Valley and at Bleiberg, the back and sides were cooled by circulation of the blast through the hollow castings of which they were formed. The capacity of this hearth was somewhat greater than that of the simple Scotch hearth, but the hot blast caused a greatly increased volatilisation loss, and on this account it has entirely dropped out of use.

3. **The Water-cooled Hearth.\***—This is of American origin, and its construction is seen in Figs. 21 and 22. The hearth box, *a*, filled with molten lead, upon which floats the charge of ore and charcoal, *b*, is of cast iron set in brickwork, *c*, and its capacity is about 2,500 lbs. of lead. The workstone, *d*, is a separate casting with a groove leading to the kettle, *e*. Above the hearth box is the water box or "tuyere plate," *f*, of cast iron  $1\frac{1}{2}$  inches thick, which forms the back and sides of the hearth, and in which three tuyere



Figs. 21 and 22. —Water-cooled Hearth.

holes, *g*, are cast. The water enters at *h*, and leaves at *i*. At the back is the wind box, *j*, into which the blast enters at *k*, passing into the furnace through three wrought-iron tuyere nozzles from 1 to  $1\frac{1}{2}$  inches diameter inserted in the tuyere holes. The tuyere level is from 1 to 3 inches above the constant level of the molten lead. Above the hearth is placed a sheet-iron hood (not shown) to carry off the fumes. Wood or charcoal may be used as fuel, but charcoal is more usually employed, and the capacity of each furnace is  $4\frac{1}{2}$  short tons (=4 tons English) per 24 hours. These hearths are in common use in Missouri for treating the pure non-argentiferous galena of that region, the average composition of which is PbS 97.05 per cent., ZnS 1.41 per cent., FeS<sub>2</sub> 0.34 per cent., insoluble 0.61 per cent., Ag  $1\frac{1}{4}$  ozs. per ton. It is not easy to see, however, in what way they are superior to the Scotch hearths built up of heavy castings cooled by radiation, except in being somewhat cooler for the men to work at. The capacity is no greater, and, on the other hand, the cost of replacement of the iron water-back (made in one piece) through cracking and wear and tear, is probably over a series of

\* Broadhead, *Geol. Survey of Missouri*, 1873-74, p. 492; Williams, Industrial Report, *ibid.*, 1877, p. 36; also *Trans. A.I.M.E.*, vol. v., p. 324.

years not less than that of the heavy Scotch castings, which are reversible as they become burnt, and have four wearing faces. The scarcity and cost of water would also be in many cases a difficulty, and an objection. The total weight of the Scotch hearth is considerable, but the pieces composing it are, with the exception of the hearth bottom, readily portable; and for transport in mountainous districts the latter might be, if necessary, made of wrought-iron, reinforced at the front and back with replaceable cast-iron plates to resist burning.

At the *Federal Smelting Works*,\* Alton, Ills., there are 20 water-cooled Scotch hearths of the type just described, each of which has a sump 4 feet  $\times$  2 feet  $\times$  1 foot deep, with cast-iron water jackets for back and sides having a water-space only 1 inch wide, and each has eight pipe tuyeres, 1 inch diameter, entering 2 inches above the surface of the bath of lead. Coke is used as fuel, the consumption being 8 per cent. by weight, and each furnace is worked on eight-hour shifts by two men per shift who are paid by piecework; the quantity of lead produced from 80 per cent. ore being about 10 short tons per 24 hours. Over each furnace is an iron hood, which draws off the fumes and delivers them through a collecting pipe to a dust-flue 1,500 feet in length and 10 feet  $\times$  10 feet in cross-section. A fan at the end of this having a capacity of 100,000 cubic feet per minute forces the fumes into the bag-house, which contains 1,500 bags of cotton cloth, 25 feet long by 18 inches diameter, in which the fume is collected. The lead from the hearths is run into 35-ton kettles, and refined by passing steam through from 30 minutes to one hour, after which it is cast into bars, which carry from 0.05 to 0.1 per cent. copper and traces of cobalt and nickel. The direct yield of lead in the hearth from 80 per cent. ore is said to be 70 per cent, which doubtless means that proportion of the total present, or a recovery of 56 per cent. on the weight of ore.

At the *Petraeus Co.'s Works*† four single so-called "Jumbo" hearths are in use, each of which, worked by two smelters, one moulder, and a yardman, smelts in a seven-hour shift 7,000 lbs. of 77 to 82 per cent. galena ore with 1 bushel of coal and 1 bushel of lime, the direct recovery in pig lead being over 60 per cent. The lead is moulded into pigs, and then re-melted in a refining kettle of 7,000 lbs. capacity, in which it is purified by poling, and then again moulded for market.

4. **The Jumbo or Moffet Hearth.**‡ The double form of this hearth, designed by Moffet, is in use at the works of the Picher Lead Company at Joplin, Mo., and its construction is shown in Figs. 23 to 26. The hearth box is not set in brickwork, but consists of a simple iron trough 4 feet long, 3 feet 6 inches wide, and 10 inches deep, divided longitudinally by a hollow partition, and supported 26 inches above the floor by means of four short cast-iron pillars, so as to cool the lead by the circulation of air underneath. Upon each long side of the trough stands a water box 6 inches wide and 20 inches high, while on each end is placed an air box 4 feet 4 inches long, 6 inches wide, and 20 inches high. The hollow partition in the centre is also carried up 20 inches high by means of a pair of long hollow boxes, each 4 feet by 7 inches wide by 10 inches high. The upper one is the air box, and is

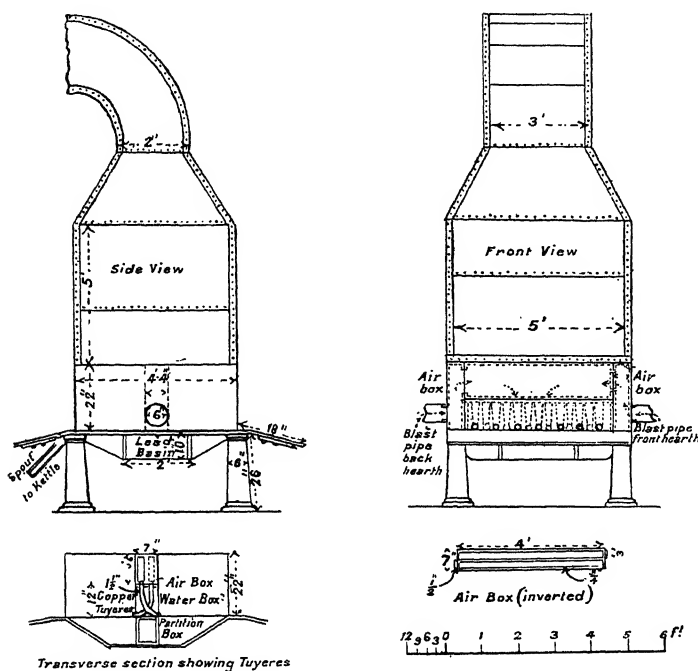
\* *E. and M. J.*, June 2, 1906, p. 1034.

† *E. and M. J.*, Nov. 9, 1907, p. 867.

‡ Dewey, *Trans. A.I.M.E.*, vol. xviii., p. 674.



partitioned longitudinally, each half being connected with the air box forming one end of the hearth through which the blast passes before entering the central air box. From each half of the latter seven copper tuyeres, each of 1 inch inside diameter, descend through the water box and deliver the blast at the level of the top of the hearth trough. The furnace thus consists of two long hearths placed back to back, water-cooled in their hottest part, and air-cooled above, provided with a hot blast so as to volatilise a large proportion of the lead, and delivering their fumes into a common sheet-iron hood 5 feet by 4½ feet by 11 feet high, which is connected with flues and dust chambers. Through these a strong current of air is drawn by means of a fan, which also forces the furnace gases first through sheet-iron cooling pipes



Figs. 23 to 26.—The Jumbo, Double, or Moffet Hearth.

and then through a series of woollen bags, in which practically the whole of the fume is deposited as a fine blue powder. The arrangement of the bag and of the bag house is similar to that of the paint room for refined fume subsequently figured and described.

Along the front of each furnace is an iron apron or workplate, and the lead overflows from one corner of the hearth into a casting-pot (not shown in the figures), set in a cylinder 31 inches diameter and 44 inches deep, and kept hot by a small fire of wood. The molten lead is drawn off by a pipe and stopcock into a row of moulds placed on a truck below, the pigs of lead weighing 84 lbs. each. The fuel is inferior bituminous coal, the consumption being 8 per cent. of the ore, and the quicklime used is about 2 per cent. of

the ore. Each fire is worked by two men on each eight hours' shift, so that there are twelve smelters required for the whole furnace in 24 hours. The daily capacity depends chiefly upon the endurance of the men, and varies from 24 short tons (of 2,000 lbs.) in summer to 30 short tons in winter. About 68 per cent. of the lead present is converted at once to metal, 22 per cent. going into the slag, and, owing to the hot blast, no less than 10 per cent. is converted into fume; the loss in condensing this fume by the bag process employed is practically *nil*.

The furnace is worked as follows\* :—After throwing on a charge the fire remains undisturbed for one to one and a-half minutes, during which time lead is reduced and the mass becomes crusted over on top. One of the two smelters then plunges a pointed bar into the melted lead at one end of the fire, and using the edge of the trough as a fulcrum prises up the mass, raising up cakes of slag and letting down undecomposed ore and fuel. In this way he works right to the other end of the fire, while the second man, following with a long-handled shovel, picks out the lumps of glowing fuming slag, which are first laid on the workplate to drain, and then tossed into a water box to cool. When the fire has been worked through by the first smelter with a pointed bar he takes the shovel, while the second man works through the fire again with a broad paddle. They then both use shovels to pick out as much slag as can be seen, after which they scatter over the fire a new charge, consisting of two shovelfuls of coal and five or six shovelfuls of ore previously mixed with about 2 per cent. of lime. The operation of working and charging takes about two minutes, and the interval of rest between each operation is about one and a-half minutes. During this interval the tools are left resting on the workplate with their handles in swing hooks, while the ore oxidises and is reduced to metal. As already mentioned, 68 per cent. of the lead present is obtained directly in pigs, and as the ores contain usually about 73 per cent., this amounts to about one-half the weight of ore treated. Most of the zinc in the mineral goes into the slag, together with 22 per cent. of the lead.

It is obvious that the working of the Moffet hearth is cheaper than that of any other form of hearth; but unless it is intended to make pigment (which can only be successful under exceptional circumstances, the market being limited) the ordinary hearth is more advantageous, as it treats in proportion to its size almost as much ore, and yields a larger proportion of lead as pigs.

**Comparison between Reverberatory and Hearth Methods.**—The advantages of hearths over reverberatories are :—(1) Smaller consumption of fuel for given capacity; (2) less first cost and greater facility for repairs; and (3) convenience which they offer for interrupted work and short campaigns. Their disadvantages are :—(1) Necessity of machinery for producing a blast; (2) larger amount of labour required and its comparatively skilled nature; (3) greater liability of the men to lead poisoning; and (4) much higher volatilisation losses. It will be obvious that the hearth process is most suitable in out-of-the-way places, where labour is cheap and fuel dear, especially if water power can be obtained for the blast, and where the supply of ores is irregular; it is only under exceptional circumstances to be recommended in localities of cheap fuel, dear labour, and regular ore supply. Galenas rich in silver should never be treated by the hearth process,

\* Dewey, *loc. cit.*, p. 682.

as the losses by volatilisation are so high; at Bleiberg (Carinthia) the average loss of lead was found to be 2 per cent. higher than in the Carinthian furnace.

**Treatment of Bye-Products.**—The bye-products of hearth smelting differ in no marked degree from those of the reverberatory; they are two—namely, grey slag and fume. The former, like the reverberatory slag, contains a large proportion of metallic lead mechanically intermingled with the mass of silicates, sulphates, oxides, and unaltered sulphides forming the true slag. Grey slag is now always smelted in slag cupolas. Fume was formerly smelted by itself in the ore hearth after thorough wetting down (and sometimes after previous roasting). More lime is generally used than in ore-smelting, and fluorspar, if obtained conveniently, is also frequently added. The consumption of coal and of labour are both greater, and the cost is consequently higher. According to Middleton,\* an ore hearth which brings out  $4\frac{1}{2}$  tons of lead per 24 hours with three eight-hour shifts of two men each, brings out only  $1\frac{1}{2}$  tons of lead in the same number of one-man shifts when smelting fume. The manipulation is similar to that when smelting ore, except that the mixture of fume with lime is kept very damp, and is banked up chiefly in the front of the fire to prevent it from being carried away by the blast. The same author gives the cost of smelting fume per ton of lead obtained as follows:—Smelters' wages, 11s. 10d.; coal (1,400 lbs.), 8s. 9d.

More frequently, however, slag and fume are smelted together in a low blast furnace provided with a firebrick lining, brasque bottom, and external iron crucible or kettle, and blown by means of one or two tuyeres at the back. The products are an impure lead, "slag lead," which is hard even after poling, a very dark "black slag," which is thrown away, and a good deal of fume, most of which is generally lost. The old English slag hearth is well described by Percy;† but as this appliance has been mostly replaced by more modern cupolas and blast furnaces similar to those employed in smelting low grade lead ores (*v. Chap. viii.*) it is unnecessary to enter into details here.

At *La Tortilla Works* (Linares‡) the grey slag from 24 of the Scotch hearths, a description of which has been already given, is smelted in two slag furnaces like those described in Chapter viii., the working column being 2 metres high, a ferruginous calcite found as lode-filling in the mines being used as flux. Each slag furnace puts through 35 tons of charge per day with a fuel consumption of only 7 per cent. The composition of the black slag made has been already given in Table VII.

At the *Petraeus Works* (Joplin)§ grey slag and fume from ore hearths are smelted in a blast furnace with metallic iron and fluxes, the following being the quantities of charge smelted per 24 hours:—Galena with 77 to 82 per cent. of lead 4,500 lbs., grey slag with 35 per cent. lead 6,000 lbs., blue fume with 72-75 per cent. lead 7,500 lbs., "carbonates" with 72 per cent. lead 500 lbs., "ashes" or coarse flue-dust with 56 per cent. lead 1,500 lbs., foul black slag with 0·7 per cent. lead 15,000 lbs., cast-iron scrap 600 lbs., wrought-iron scrap 600 lbs., iron cinder 300 lbs., and tin cans 300 lbs. This quantity of material is smelted in eight hours, and the average product in pig lead is 11,810 lbs., or 25·7 per cent. by weight on the whole charge. The composition of the slag produced is given in Table IX. The labour required

\* *E. and M. J.*, July 6, 1905, p. 11.

‡ *Private Notes*, 1907.

† *Metallurgy of Lead*, pp. 413-417.

§ *E. and M. J.*, Nov. 9, 1907, p. 867.

per eight-hour shift is one feeder at 10s. 6d., one tapper at 8s. 10d., and four yardmen at 8s. The blue fume from the blast furnace, like that from the ore hearths, is drawn through a bag house and there collected.

At the *Federal Smelting Works* (Alton, Ills.)\* the grey slag and fume from the ore hearths (the latter of which is always previously burned in order to sinter it and convert it into lead sulphate) are smelted together with roasted matte from a previous campaign in a 12-tuyere cupola provided with a lead well at the back; slag and matte are tapped together at the front, and run through a series of slag pots for separation, the shells being saved for re-smelting, while the waste slag contains only about 0.5 per cent. lead. The slag-lead from this furnace, being impure, is refined by liquation in a small reverberatory furnace with an inclined hearth of perforated plates, through which the lead runs, while the dross remaining on the plates is returned to the blast furnace. The liquated lead runs to the refining kettle, where it is "steamed" for about an hour, and after skimming is then cast into bars with a Steitz syphon (for which see Chap. viii.). The matte is crushed and roasted in a 60-foot reverberatory furnace.

### The Lewis and Bartlett Bag Process.

This process, which is in use at the Lone Elm Works of the Picher Sublimed Lead Company, and also at those of the Bristol Sublimed Lead Company, is peculiar in its aim to produce as much fume as possible; by the use of special fume filtering apparatus, and by taking special precautions in the smelting, more than one-half of the lead contained in the grey slag and flue dust from the ore hearths is worked up into a fine white pigment. The fine flue dust and fume "blue powder" from the ore hearths, owing partly to the carbonaceous and bituminous matter and partly to the finely-divided metallic sulphides contained in them, burn readily when piled in heaps on the ground and ignited by means of oiled waste, and after a few hours' smouldering are converted into a mass of pinkish-white crusts. Analyses of raw and roasted "blue powder," as well as of flue dust from the first dust chamber and of grey slag from the Moffet hearth, are given in Table IX. The grey slag contains 21.45 per cent. of metallic lead which stays on a 60-mesh sieve, and the remaining portion, according to the analysis given in the Table, contains 47.45 per cent. of lead in combination, making a total of 58.71 per cent. in all.

The modified slag hearth or "slag-eye furnace," in which the grey slags and blue powder are worked up together, is 2 feet square by 6 feet high altogether, the distance from lower tuyeres to the charging door being only 2 feet 3 inches, and the charging door 4 feet from the ground. Its construction will be understood by reference to Figs. 27 and 28.† Blast is supplied by means of eleven 1½-inch tuyeres passing through water-boxes; four of them are at the bottom for fusion and reduction, the remaining seven being only a few inches below the charging door, in order to secure as much volatilisation as possible. The furnace is kept filled to the charging door with a bed of glowing coke, upon which the charges are spread. All the bye-

\* *E. and M. J.*, June 2, 1906, p. 1034.

† Dewey, *Trans. A. I. M. E.*, vol. xviii., pp. 683 and 689; more recently this furnace has been improved by replacing its brick walls with water-jackets.

# THE METALLURGY OF LEAD.

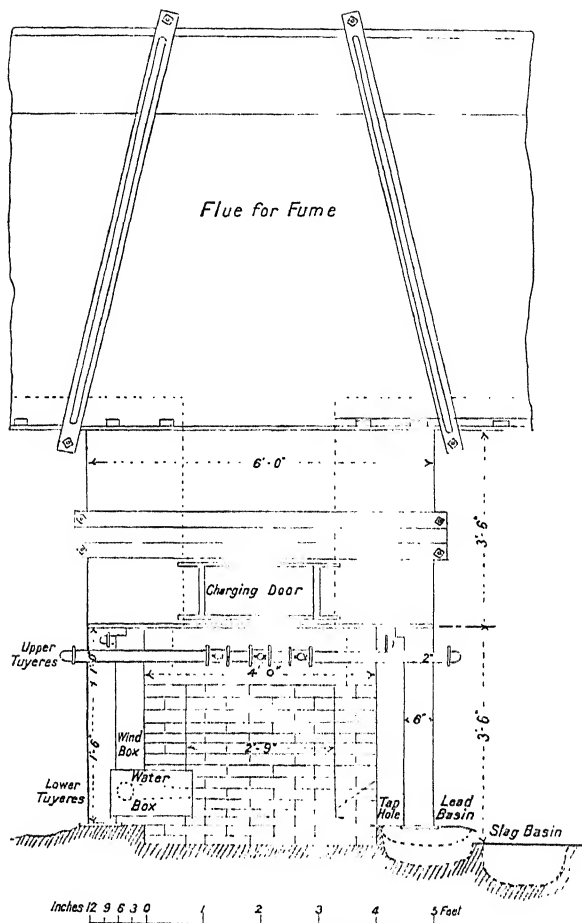


Fig. 27.—Lewis and Bartlett Process—Slag Hearth—Elevation.

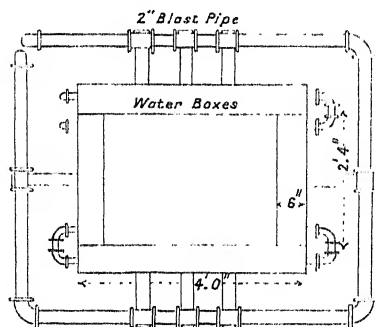
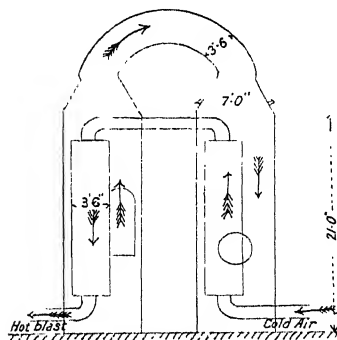


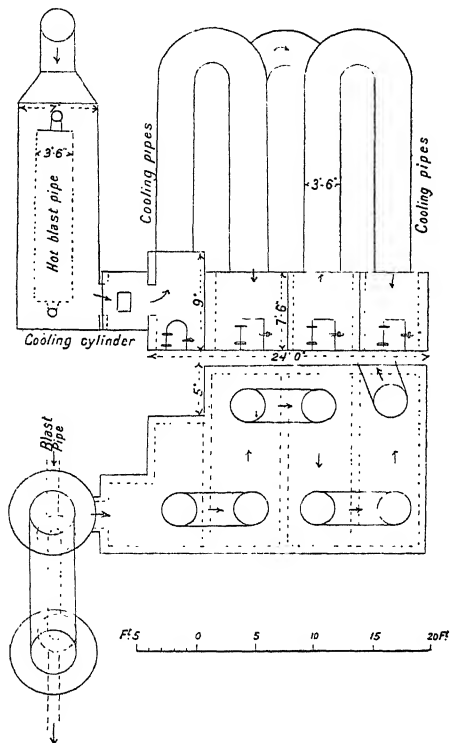
Fig. 28.—Section through Upper Tuyere.

products of the ore hearth are treated in this furnace together with a small amount of oxidised ore. An average daily charge for one furnace is made up as follows:—Grey slag, 2,800 lbs.; “blue powder,” 1,000 lbs.; “dry bone” (oxidised lead and zinc ores), 600 lbs.; half-refined blue powder from cooling pipes, 450 lbs.

The object being to volatilise lead rather than to reduce it, the top of the furnace is run so as to keep the flue nearly red hot. It is found that a hot flue is necessary for the production of a good white colour in the pigment, and with the same object only hard Connellsville coal is used as fuel. The waste heat in the furnace gases is made use of in order to heat the blast, which also assists in volatilising lead. In spite of all endeavours a considerable proportion of lead is, however, reduced, and this, together with the poor “black” slag formed, finds its way through a hole in a “tap-plate” at the bottom into an iron “lead basin” with two unequal compartments, the larger being filled with charcoal. The slag flows over the charcoal into the “slag basin” placed below,



Section of cooling cylinders.



Fume cooling pipes and cylinders.

Figs. 29 to 31.—Lewis & Bartlett Process—Coolers.

through which a stream of water flows, while the lead filtering through the charcoal passes under the partition into the smaller compartment, whence it is ladled into moulds. This lead is of very inferior quality, and has to be remelted and poled to fit it for sale; about 13 pigs, or 1,100 lbs., are produced on an average daily. The slag is nearly black, and, though still carrying a fair percentage of lead, is thrown away, except a small proportion which is thrown into the furnace against badly corroded places in the walls

as a partial protection.\* Dewey gives an analysis showing 23 per cent. of  $\text{PbO}$  in this slag, but at a later date it was reported to contain under 10 per cent.†

The furnace is started by lighting a fire on the bottom and filling the shaft with charcoal free from breeze, after which a light blast is started; when the charcoal is glowing, a thick bed of coke is charged and the full pressure of blast (9 ozs.) turned on. As soon as this is thoroughly ignited on top the ordinary charge is given, each charge consisting first of a bed of coke, distributed chiefly towards the front and centre of the shaft, then of slag which is heaped against the back and sides so as to protect them some-

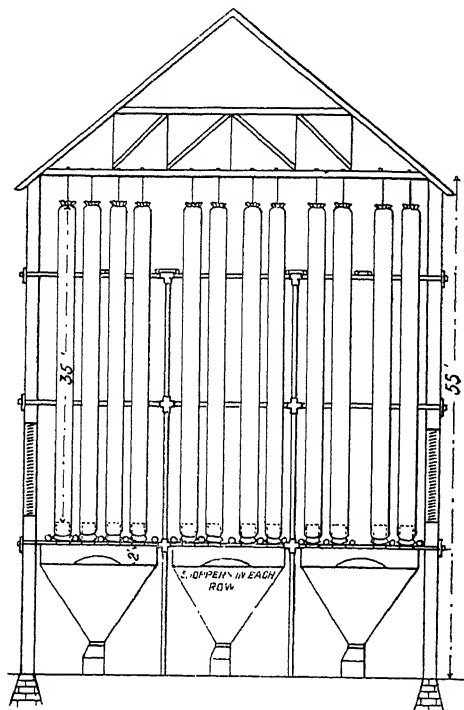


Fig. 32.—Lewis & Bartlett Process—Bag-house.

what, and, lastly, the blue powder. Two men attend the furnace on each eight-hour shift, one to feed, and one to ladle lead and wheel slag.

A Baker blower (No. 5) supplies air through an 18-inch main to three of these "slag-eye" furnaces, two of which are run together, the third being in reserve. From the flue a fan (6 feet diameter and 3 feet face), run at 290 revolutions per minute, draws the fumes and gases, first, through a pair of iron cylinders 7 feet diameter and 20 feet high, lined with firebrick, through which the 18-inch blast main, temporarily enlarged to 3 feet, passes in order

\* *v.* Note on previous page as to use of water-jackets.

† *E. and M. J.*, Dec. 8, 1894.

to heat the blast, and, secondly, through a series of four inverted U-pipes,  $3\frac{1}{2}$  feet diameter and 20 feet high, set in brickwork and exposed to the air. These cooling arrangements are shown in plan and elevation in Figs. 29, 30, and 31. Some fume settles out in the pipes, but it is of poor colour, and is, therefore, returned to the furnace. After passing through the cooling pipes the fumes enter the white bag room, or "paint house," which is 90 feet by 40 feet by 45 feet high in two storeys with a boarded floor, the arrangement of which may be seen from Fig. 32. There are three rows of hoppers, nine in each row, and above each hopper are suspended twenty bags of coarse flannel made from unscoured wool, 20 inches in diameter and 35 feet long, costing \$9 apiece, and tied over nipples projecting from the top of the hopper. The bags are suspended from the rafters, the hoppers and flues are also suspended by means of iron straps from a skeleton flooring of  $1\frac{1}{2}$ -inch pipe laid 2 feet apart across a framework of  $2\frac{1}{2}$ -inch pipe, which is itself supported by eighteen columns of  $3\frac{1}{2}$ -inch pipe. Each bag has 168 square feet surface, so that the filtering surface of the whole 540 bags is 88,704 square feet.

The paint from different parts of the room is of about the same quality, and has good body and colour. It is packed in barrels holding 500 lbs. each, and sells at a slightly higher price per lb. than pig lead, 1 lb. of which makes 1.6 lbs. of paint. The daily product of each furnace is about 4,250 lbs.

Analyses of the white paint as well as of the various intermediate products of the process are given in the following table:—

TABLE IX.\*

	Grey Slag.	Chamber Dust.	Flue Dust.	Blue Powder.		White Paint.	Black Slag.†
				Raw.	Roasted.		
SiO <sub>2</sub> , . . . .	12.70	2.76	2.54	0.12	0.10	0.08	27.5
FeS <sub>2</sub> , . . . .	0.67	...	...	...	...	...	...
ZnS, . . . .	3.64	5.57	0.11	...	...	...	...
PbS, . . . .	14.73	51.59	10.41	7.59	...	...	...
PbSO <sub>4</sub> , . . . .	5.18	22.74	61.39	45.08	48.76	65.23	...
PbO, . . . .	33.55	3.79	11.47	44.55	46.82	25.87	4.0
ZnO, . . . .	14.53	0.35	0.42	0.42	0.27	5.98	8.5
MnO, . . . .	...	...	...	...	...	...	...
FeO, . . . .	...	...	...	...	...	...	33.5
Fe <sub>2</sub> O <sub>3</sub> , . . . .	2.90	0.93	0.97	} 0.07 {	0.32	0.03	...
Al <sub>2</sub> O <sub>3</sub> , . . . .	0.78	0.06	0.05		0.05	...	...
CaO, . . . .	11.49	6.40	5.24		0.48	0.02	22.0
MgO, . . . .	0.12	0.03	0.03	...	...	...	...
CO <sub>2</sub> , . . . .	...	3.83	1.35	0.19	0.90	1.77	...
SO <sub>2</sub> , . . . .	...	0.86	...	0.55	1.65	0.02	...
H <sub>2</sub> O, . . . .	...	...	...	...	0.37	0.72	...
Org. matter yielding:							
CO <sub>2</sub> , . . . .	...	14.41	7.38	1.93	...	...	...
H <sub>2</sub> O, . . . .	...	3.38	3.53	3.18	...	...	...
...	...	...	...	...	99.72	99.70	...

The bag process for the collection of fume has now come into pretty

\* From Dewey, *loc. cit.*

† Hofman, *Met. of Lead*, 1906, p. 130.



general use throughout the soft lead regions of Western America, although in consequence of the small demand for white fume prepared by the Bartlett process nearly all the fume collected has to be re-smelted.

At the *Petraeus* Co.'s Works\* the bag-house is 100 feet  $\times$  68 feet, and contains 680 bags, each 30 feet long  $\times$  2 feet diameter, made of cotton cloth with 44 threads to the inch. The fumes from four single Jumbo hearths and a blast furnace are forced through these bags by a 90-inch Buffalo fan, after passing through a "trail" flue 6 feet  $\times$  4 feet in the clear and 800 feet long.

\* *E. and M. J.*, Nov. 9, 1907, p. 867.

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## CHAPTER V.

## ROASTING LEAD AND SILVER ORES AND MATTES.

**The Roast-Reduction Process.**—The roast and reduction process is adapted to the treatment of ores with any kind of metallic and non-metallic admixture; siliceous ores, and those containing arsenic and antimony being always treated in this way. It comprises the separate operations of *roasting* and *smelting*.

Nearly all works employing the roast-reduction process have to treat, in addition to lead ores proper, other ores which contain a greater value in the precious metals than in lead, and in some cases contain no lead at all. As the treatment of such ores has to be undertaken in connection with that of lead ores properly so-called, and since in practice it is generally found convenient to mix leady ores and mattes with non-lead sulphide ores before roasting, in order to facilitate the operation by diminishing the tendency to sinter, it is necessary to consider the roasting of lead ores as including also the roasting of all ores and mattes which have to undergo an oxidising operation preparatory to smelting in a lead furnace.

The object of roasting is to drive off as much as possible of the sulphur (as well as arsenic and antimony, when present), and convert the lead as far as possible into oxide. Some lead sulphate is always formed, but the amount of this substance is kept as low as possible, since, unless it should happen to come in contact with and react upon unaltered sulphide, it might be itself reduced to sulphide. In many cases towards the end of the operation, the temperature is raised in order that the silica may react upon the sulphate, forming silicate, and at the same time partially fusing the material so as better to fit it for smelting.

In the ordinary smelting process conducted in blast furnaces the lead is reduced from its oxide partly by means of carbonic oxide, but chiefly by direct contact with masses of glowing, porous fuel (coke or charcoal). Such lead sulphide as may remain in the roasted ore or become reduced from the sulphate is also decomposed by the action of iron reduced from its oxide, the lead being liberated while the sulphur forms an *iron matte*. Other metallic constituents of the ore, except gold and silver, which are collected in the lead, pass either into the slag or the matte, or become volatilised.

**Reactions of the Roasting Process.\***—As we have seen, therefore, lead ores, which have to be roasted preparatory to smelting, are more or less finely pulverised mixtures containing, besides galena, varying proportions of blende, pyrites, chalcoppyrite, quartz, and other gangue materials. Table X. gives analyses of a few of the complex lead sulphide ores which are subjected to roasting processes.

\* Roberts-Austen, Pres. Address, British Assoc. Meeting at Cardiff, 1891; also Rose, *Metallurgy of Gold*.

TABLE X.—ANALYSES OF COMPLEX LEAD SULPHIDE ORES.

	Przbram, Bohemia.	Rodna, Transylvania.	Goslar, Lower Harz.	Clausthal, Upper Harz.	Mechnich, Rhenish, Prussia.	Col. Sellers, Leadville, Colorado.	Broken Hill, N.S.W.	Sunny Corner, N.S.W.
Reference,	1.	2.	3.	4.	5.	6.	7.	8.
PbS, . . . .	45.09	54.57	10	74.09	69.74	18.67	28.90	16.30
ZnS, . . . .	8.73	0.99	36	7.46	0.22	28.73	37.20	22.00
CuFeS <sub>2</sub> , . . . .	0.12	tr.	1.5	1.15	0.51	tr.	tr.	4.50
FeS <sub>2</sub> , . . . .	4.65	42.62	24	0.60	1.39	49.29	0.50	37.30
Sb <sub>2</sub> S <sub>3</sub> , . . . .	1.07	0.02	...	0.80	0.11	...	...	...
Ag <sub>2</sub> S, . . . .	0.313	0.067	...	0.10	0.017	...	0.091	...
MnS, . . . .	...	...	...	...	...	2.61	...	...
SnO <sub>2</sub> , . . . .	0.14	...	...	...	...	...	...	...
FeCO <sub>3</sub> , . . . .	13.77	...	...	2.27	...	...	...	...
MnCO <sub>3</sub> , . . . .	1.61	tr.	...	0.48	...	...	...	...
CaCO <sub>3</sub> , . . . .	1.75	tr.	...	1.71	1.57	...	...	...
MgCO <sub>3</sub> , . . . .	0.57	...	...	0.35	...	...	...	...
BaSO <sub>4</sub> , . . . .	...	...	20	...	...	...	...	...
CaSO <sub>4</sub> , . . . .	...	...	...	1.92	...	...	...	...
SiO <sub>2</sub> , . . . .	20.05	0.49	...	8.57	22.05	0.96	32.00*	16.50
Al <sub>2</sub> O <sub>3</sub> , . . . .	1.63	...	} 8.5	{ 1.32	3.60	...	...	...
As, . . . .	...	0.34				tr.	...	0.30
Ag. ozs. per ton, .	89.3	19.3	...	28.5	34	7½	21	12

NOTES.—1, 4, 5 are mixtures of ore and concentrates; 6 and 7 are natural ores which have to be concentrated before roasting; 8 was formerly roasted and smelted without concentration.

References.—1. *E. and M. J.*, July, 16, 1892. 2. Hofman, *Metallurgy of Lead*, p. 167. 3. Schnabel, *Handbuch der Metallhüttenkunde*, vol. 1., p. 389. 4. *E. and M. J.*, Oct. 21, 1883. 5. Schnabel, *op. cit.*, p. 393. 6. Taylor and Brunton, *E. and M. J.*, May 8, 1886. 7 and 8. Author.

The reactions which take place in roasting such mixtures are as follows :—

The first action, as the temperature of the charge rises, is the distillation of about one-third of the sulphur in the pyrites, which burns with a blue flame to SO<sub>2</sub>, part of which again in contact with oxygen and with the hot porous charge and furnace brickwork is converted into SO<sub>3</sub>. The next action is the successive oxidation of the metallic sulphides by the current of hot air, approximately in the following order, Fe<sub>3</sub>S<sub>4</sub>, CuFeS<sub>2</sub>, Ag<sub>2</sub>S, Cu<sub>2</sub>S, PbS, ZnS. Of these, *silver* sulphide is soon reduced to metal, and would remain so if it occurred alone, but the presence of much base metal sulphides and excesses of free SO<sub>3</sub> enables it to be converted into sulphate; *iron* and *copper* sulphides are converted, with free access of air, almost entirely into oxides; *lead* and *zinc* sulphides partly to sulphates, but chiefly into oxides, free SO<sub>3</sub> in the vapours, however, reconverts varying proportions of the oxides of all these last four metals into their respective sulphates.

\* This is not silica, but a mixture of silicates, including rhodonite, garnet, felspar, &c., as well as quartz.

This process of oxidation is not continuous, except in a strong draught, for as soon as the sulphurous acid reaches a certain tension chemical equilibrium is established and oxidation is arrested, even in presence of an excess of oxygen, unless the temperature be raised considerably, so as to destroy equilibrium.

Besides a temperature which should be easily controlled, efficient oxidising roasting, therefore, requires abundant access of air to the hot ore and rapid removal of the products of combustion. If the temperature be kept moderately high and the draught and supply of air low, most of the lead and zinc will be converted into sulphates, partly by direct action of  $\text{SO}_2$  and O, partly by the  $\text{SO}_3$  evolved from decomposing iron sulphates. The temperature must, in any case, be well under control, for, if raised too suddenly, portions of the ore will undergo incipient fusion, and particles of unaltered sulphides be protected by an envelope impervious to the further action of the gases.

As the temperature rises, and before the oxidation of the sulphides is complete, the sulphates formed begin to dissociate in the following order:— $\text{Fe}_2(\text{SO}_4)_3$ ,  $\text{FeSO}_4$ ,  $\text{CuSO}_4$ ,  $\text{Ag}_2\text{SO}_4$ ,  $\text{ZnSO}_4$ . Iron and copper sulphates give off, at first,  $\text{SO}_3$ , and, afterwards,  $\text{SO}_2$  and O; silver sulphate remains undecomposed until all the iron and nearly all the copper sulphates have been decomposed, when it in turn breaks up into metallic silver, oxygen, and  $\text{SO}_3$ ; zinc sulphate is nearly unchanged in simple roasting, giving off only part of its  $\text{SO}_3$  at a full red heat, the basic sulphate remaining not being decomposable under a bright yellow heat. Lead sulphate is unaltered.

Among gangue substances quartz and barytes remain unaltered, *chalybite* is converted into magnetic oxide of iron,  $\text{Fe}_3\text{O}_4$ , and *calcite* into calcium sulphate by expulsion of  $\text{CO}_2$  and absorption of O and  $\text{SO}_2$ .

As regards removal of the deleterious substances, arsenic and antimony, it may be remarked that *mispickel* ( $\text{FeSAs}$ ) is pretty thoroughly decomposed, the greater part of its arsenic being volatilised direct as  $\text{As}_2\text{S}_3$ , which is oxidised into  $\text{As}_2\text{O}_3$  and  $\text{SO}_2$  by the current of hot gases, the remainder being oxidised direct to  $\text{As}_2\text{O}_3$  at a low temperature in presence of a good draught.

Most *sulph-antimonides* and *sulph-arsenides* (e.g., fahlerz, bournonite, &c.), like mispickel, give up the greater part of their arsenic (antimony) as  $\text{As}_2\text{S}_3$  ( $\text{Sb}_2\text{S}_3$ ), which are at once oxidised to  $\text{As}_2\text{O}_3$  ( $\text{Sb}_2\text{O}_3$ ) and  $\text{SO}_2$ . Another portion of the arsenic (or antimony) is converted into arseniates (or antimoniates), which resist the effects of simple heating. These compounds are, however, reduced in contact with metallic sulphides, especially with free pyrites, and, therefore, when the charge is rabbled the arseniate (or antimoniate) formed in the oxidised surface crust gets turned over into the bottom layer of the ore, where there is more or less free sulphur, and is re-volatilised to the surface as  $\text{As}_2\text{S}_3$  ( $\text{Sb}_2\text{S}_3$ ), so that it gets another chance of escape. *Enargite*, however, is only partially decomposed, most of the arsenic remaining behind as arseniate. It is noteworthy that, whereas the ferric arseniate produced in small quantity by the oxidation of mispickel is comparatively easily decomposed by means of heat and reducing agents (carbon or sulphur), *cupric arseniate* formed by partial oxidation of enargite is most obstinate in resisting decomposition; in fact, according to Pearce,\*

\* *Trans. A.I.M.E.*, vol. xviii., p. 62.

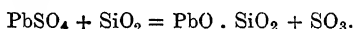
scarcely any of the arsenic of enargite can be driven off by roasting, even when assisted by the subsequent rabbling in of fine coal.

Simple *arsenides* and *antimonides* (e.g., smaltite, domeykite, dyscrasite) are much more difficult to decompose, by far the larger part of their arsenic remaining as stable arseniates and antimoniates of the metals present (iron, copper, and silver). According to Howe and Campbell,\* only 15 per cent. of the arsenic present is expelled below 700° C., and an additional 24 per cent. below 890°, above which point the further expulsion of arsenic is very slow, even if charcoal be added to decompose the basic arseniates formed. Such minerals, therefore, like the *sulph-arsenides*, should, whenever possible, be roasted together with an excess of sulphides, such as pyrites; use is also made of this reaction of sulphur upon combined arsenic in the treatment of arsenical furnace products.

That a large part of the antimony, as well as arsenic, which is expelled from antimonial and arsenical ores during roasting escapes as sulphide is proved by the fact, noted by Wendt,† that in kiln roasting lump ore, crystals of artificial *stibnite*, as well as of *orpiment*, were found in the flues and on the upper and cooler portions of the charge. The occurrence of *orpiment* and *realgar* on the upper surface of roast heaps is a matter of every day observation.

The temperature reached in ordinary roasting for the elimination of sulphur without sintering are only from 900° to 1,000° C.‡

**Sinter- and Slag-Roasting.**—When at the end of the roasting proper the temperature is suddenly raised so as to agglomerate or fuse the charge, the lead sulphate formed during roasting is decomposed by means of silica (either already present in the ore or added, if necessary, in the form of more siliceous ores); for, although lead sulphate is practically unaltered at the temperature reached in the furnace, free silica drives out the sulphuric acid in accordance with the equation—



Even when the temperature does not rise high enough to decompose lead sulphate, there is more or less combination between silica and lead oxide, so that the mass begins to soften and agglomerate without actual fusion. At the temperature of fusion *calcium* and *barium* sulphates in the roasted ore are also partially converted into silicates, or in the case of lime to ferrites, and if silica be present in sufficient quantity the iron oxides are also converted into silicates to a greater or less extent, according to the perfection of the fusion. *Zinc* is to a small extent volatilised, but is mostly melted down, with the oxides of iron and lead, into a readily fusible double silicate.

The final result of the combined roasting and melting process is a fluid or pasty mixture of oxides and silicates, never quite free from sulphides, which is tapped out into iron moulds, and, after being broken up, forms a material better suited to blast-furnace treatment than the fine powder resulting from plain roasting.

\* *Trans. A.I.M.E.*, vol. xxxviii., p. 170.

† *Ibid.*, vol. xix., p. 100.

‡ The composition of a number of samples of roasted sulphide lead ores is given in Table xv., Chap. vii. As regards the condition of the residual sulphur which has not been oxidised to sulphate of lead, zinc, lime, &c., reference should be made to two interesting articles in the *Engineering and Mining Journal* of April 13, 1907, p. 707, and Feb. 9, 1905, p. 285, in the latter of which some evidence is adduced to show that even well roasted ore may contain S still in the condition of  $\text{FeS}_2$ .

When the ores to be fused contain less than 10 or 15 per cent. lead, the charges have to be roughly calculated, so as to contain silica and iron in suitable proportions to form a fusible slag. This is readily done, as almost every large smelting works treating custom ores will have both pyritic and quartzose ores at command. Lime is rarely added to a charge in the fusion hearth, as sufficiently good slags can always be made with iron alone and whatever lead or zinc may be present. The composition of slag-roasted ore as ordinarily made is as follows:— $\text{SiO}_2$ , 20 to 35 per cent., Fe 20 to 25 per cent., Zn 5 to 15 per cent., Pb 10 to 20 per cent., S as sulphide about 2 per cent., but sometimes the lead is higher.

The difference in the condition of a blast furnace resulting from the use of slagged or sintered sulphide ores, instead of the plain roasted fines, is remarkable. Much greater regularity in working is accompanied by much more rapid smelting, and by a smaller coke consumption, while the losses in slags and flue dust are much diminished, and the percentage of sulphur being much less, the amount of matte produced is smaller. In view of these advantages, "slagging" or fusion would be widely adopted in place of simple roasting (in spite of its greater cost both for fuel and labour), but for the heavy volatilisation losses involved. According to Newhouse,\* the average loss in plain roasting silver-bearing lead ores containing 12 to 18 per cent. lead is from 2 to 5 per cent. of the lead and hardly any of the silver, while in fusing down the same ore the loss is increased to 15 or 20 per cent. of the lead and 2 to 3 per cent. of the silver. The lead loss in simple roasting at a low temperature is chiefly as fine dust mechanically carried off by the draught and recoverable in suitable flues and settling chambers, while the loss in the fusion process is as a true fume, very little of which is recovered in ordinary flues. The losses in slightly "sintering" or "agglomerating" are not very much higher than in plain roasting, therefore rich lead ores should be only "sintered," never "slagged." With ores rich in lead the agglomeration need not be carried very far, a very slight rise in temperature above that normally employed giving the material sufficient plasticity to enable the furnacemen, by dint of ramming it into the slag pots as drawn, to produce coherent masses, which only partially break up before charging, and are, therefore, much less liable to furnace losses than mere powder. According to Hofman,† only such ores should be slagged as contain under 10 per cent. of lead and 100 ozs. of silver per ton. Since the introduction of blast-roasting methods (described in the next chapter), whereby the high temperature required for agglomeration and partial slagging is obtained, not throughout the whole charge at one time, but in successive small portions of it, and in such a way that the focus of high temperature is always surrounded by comparatively cold ore, so that any substances volatilised have a good chance of becoming recondensed in the surrounding cold charge, ordinary slag roasting, as applied to mixed pyritic ores, has become obsolete. Purely lead ores and mattes are, however, still in some places roasted and sintered in hand reverberatories.

According to Iles,‡ the loss caused by slagging roasted ore does not stop

\* *E. and M. J.*, Feb. 28, 1891.

† Discussion on "American Improvements, &c.," *Trans. A.I.M.E.*, vol. xxii., p. 657.

‡ *E. and M. J.*, vol. lxviii., Sept. 16, 1899, page 341.

at the volatilisation loss in the roaster, for the lead silicate formed by slagging is much more difficult of reduction than if left in the condition of oxide or sulphate, and so the blast-furnace slags carry more combined lead; he also claims that zinc, which might be converted into oxide by simple roasting, is often prematurely fused as sulphide in the fuse box, and must subsequently enter the matte or slag in that form, causing the troubles to which reference will be made in the following chapters.

**Roasting Appliances.**—At different localities the roasting of leady ores and mattes may be found to be carried out in either of the following types of appliance—viz., heaps, stalls, kilns, reverberatory furnaces, both hand and mechanically stirred; and even in a few cases shaft and muffle furnaces are employed.

#### ROASTING LUMP ORES AND MATTES.

**Heap-roasting** is only applicable to lump ores or to fine ores artificially made to take a lump form. Fines may be employed in certain proportions as a covering, but if it is wished to use them in large proportion they must be “bricked” or “balled” with an admixture of lime or of waste liquors containing ferrous sulphate. A single roasting in heaps is usually not sufficient, and, as a rule, several roastings are required.

In the heap roasting of mixed ores there is always a tendency for the copper (and, according to Vogt, for cobalt and nickel also) to become concentrated in a “kernel,” which resembles in composition a copper matte of 7 to 50 per cent.\* According to Plattner,† when the roasting proceeds very slowly silver and lead exhibit a well-marked tendency to pass towards the *outside* of the lump; for example, in a well-roasted piece of heap-roasted iron-lead matte the central kernel contained only 5·2 per cent. Pb and 0·075 per cent. Ag, whereas the outer envelope contained 17·2 per cent. Pb and 0·230 per cent. Ag. This is confirmed by the fact that films of metallic silver have been not infrequently found on the outside of lumps of calcined ore, where, no doubt, they had been deposited by the decomposition of some volatile compound of the metal. When, however, the roasting proceeds quickly, so that the temperature rises to the fusing point of a low-grade matte, copper, lead, and silver are all concentrated indiscriminately by a sort of liquation process into a low-grade matte which is found in the lower portions of the pile.

At Goslar (Lower Harz) heap roasting is still practised on the pyritic ores from the Rammelsberg, which carry blende and barytes, but are too poor in lead and silver for the more costly roasting in reverberatories. The ore is composed roughly as follows:—

	Per cent.		Average Composition per cent.
Galena,	9 to 10	S,	16 to 18
Blende,	27½ to 30	Ag,	0·011 to 0·015
Chalcopyrite,	1 to 1½	Cu,	0·45 to 0·55
Pyrites,	11 to 16	Pb,	9·98 to 10·5
Gangue (chiefly barytes),	44 to 47	Zn,	18 to 19
		Fe,	5 to 7

\* v. Peters, *Modern Copper Smelting*, 7th Ed., 1895, p. 134; also Schnabel, *Handbuch der Metallhüttenkunde*, vol. i., p. 36.

† *Die Metallurgischen Röstprozesse*, Freiberg, 1856, p. 205.

The floor of the roast yard is formed of well tamped clay, upon which is spread a layer of roasted fines. The heaps hold 300 to 500 tons each, and are built up on a square bed of logs and brushwood 16 to 20 inches thick and 33 to 40 feet in the side, channels being left for the air currents. Upon this bed is spread, first, the coarsest ore, which comprises the chief part of the heap; then, successively, the pieces of fist, egg and nut size; and, lastly, a layer of raw fines. The heap is then covered with a layer of sifted once-roasted fines, and forms a truncated pyramid on a square base 36 to 40 feet square at bottom and, say, 12 feet square at top. When ready, it is lit on the side opposite to the wind. In this way, since air can only enter through the channels left in the bed of fuel, during its passage through the lower part of the heap it becomes deprived of its oxygen, so that the hot gases which reach the upper part of the heap are composed principally of nitrogen and sulphurous acid. These gases volatilise from the pyrites a part of its sulphur, which condenses in a liquid form in bowl-shaped depressions about 8 inches across formed in the upper part of the heap and lined with ore fines, from which it is from time to time removed. The first sulphur appears fourteen days after lighting the heap.

The first roasting lasts from six to seven months, after which the coarse ore is screened from the fines (which are leached for zinc vitriol\* and then melted) and subjected to a second and even a third similar roasting conducted under sheds to protect it from the weather and from high winds. The second roasting lasts six to eight weeks, the third from four to six weeks. The coarse well roasted ore is then smelted,† together with the leached fines.

The roasting of 100 tons of ore requires altogether 13·9 cubic metres (3·8 cords) of billet wood and 212 faggots of brushwood. It will be readily understood that in the foregoing complicated series of operations there is great danger of losing lead and silver in fine ore dust mechanically carried away by the wind, &c.; especially as, in the roasting process, silver appears to become concentrated in the outermost thoroughly calcined shell of each lump of ore, where it is most liable to loss by abrasion, &c.—the exact opposite being the case with copper.

At *Sunny Corner* and at *Lake George* (New South Wales) complex ores, composed chiefly of pyrites, quartz, and blende, with from 5 to 10 per cent. of galena and 3 to 4 per cent. of chalcopyrite, were, until within the last few years, roasted on a large scale in heaps of from 800 to 2,000 tons, built from tramlines at the level of their tops by means of iron turntables and movable sections of track. No care was taken to separate coarse from fine, but after laying a bed of wood in loose chequer work about 15 inches deep, a layer of selected large lumps of ore was first spread over it, and then the "run of fine" ore deposited upon it exactly as trammed from underground, upright logs of wood being inserted into the roast bed at regular intervals a few yards apart to serve as chimneys. With such highly pyritic ore the quantity of wood used per ton of ore is small, and its cost but a few pence per ton, since wood is only required for ignition purposes, and the ore once ignited burns readily and thoroughly. A heap of ore can be thoroughly burnt out in about two to three months if left alone, but by commencing to remove the burnt ore as soon as the outer fringe is properly roasted, the whole heap can be burnt out and removed in a month when not over 7 or 8 feet high.

\* *v.* Chap. xx.

† *v.* Chap. xi.



All the objections to heap-roasting of ores apply with even greater force to the roasting of *matte* in heaps, while the product is usually not so well roasted, nevertheless the method is still employed to some extent.

At *Clausthal* (Harz) *matte* is roasted in heaps of 150 to 200 tons built on a bed of cordwood, with a second bed of brushwood and coal-slack between the largest lumps of *matte* ranged above it. The first burning lasts two to three months, and yields about one-third as well-roasted product carrying about 6 per cent. sulphur, principally as sulphates. The remaining two-thirds require two or three more burnings in smaller heaps of 75 to 100 tons, built up as before, the well-roasted product being hand-picked, and the residue passed on to another heap.

At *Córdoba*, *Peñarroya*, and other places in Spain, lead mattes are roasted in small heaps of 40 to 50 tons by means of screened cinders from reverberatory furnaces or from steam boilers, layers of fist-sized lumps of *matte* alternating with layers of screened cinders 2 inches thick until the pile is completed. Such small piles burn out in three to four weeks, and any tendency to over-heating and fusion is checked as soon as noticed by a layer of fine ashes over the part affected.

In *Utah* and other parts of the United States heap-roasting was commonly practised up to within a few years. The piles were usually made quite small, measuring 24 by 18 feet in the base and, say, 6 feet high, on a bed of light wood from 6 to 15 inches thick. They held about 80 tons each, and burned from thirty to forty days. Fine *matte* was kept to spread over the heap and to check combustion in case the pile should get too hot, but in spite of this, fusion or, rather, liquation of the lower part of the pile frequently took place. Terhune mentions\* a case where the fused "bottoms" thus produced assayed twice as high in lead and silver as the original *matte*. He gives the cost of roasting down to an average of 6 per cent. sulphur, as \$2.25 (9s. 4d.) per ton, made up of labour (breaking, transportation, building, and breaking down piles) \$1.65 (6s. 10d.) and wood (one-seventh cord) 60 cents (2s. 6d.) per ton.

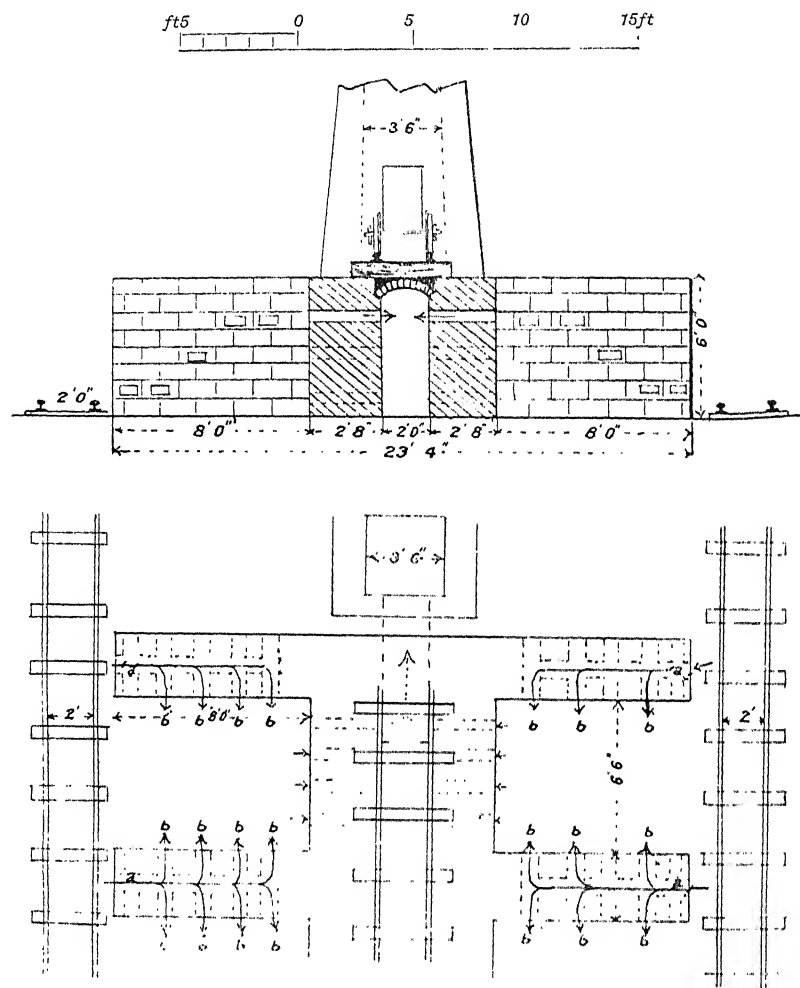
The only advantages of heap-roasting are cheapness and absence of expenditure on plant. Its principal disadvantages are:—(1) Slowness of roasting, which means locking up capital for months; (2) imperfection of roast, with the consequent necessity of re-handling and re-roasting a considerable part of the whole; (3) great losses of valuable metals through "dusting" and leaching out of silver and copper as sulphates.

Stall-roasting possesses the same disadvantages as heap-roasting, but in a less degree. Although some expenditure on plant is required, it is much less than in the case of furnace-roasting; while, compared with heap-roasting, the roast is quicker and more perfect, the losses by dusting and leaching are smaller, and the fumes, even if they are not utilised, can be rendered less obnoxious. The operation is very similar to heap-roasting, except that on account of the steadier draught the process goes on somewhat quicker; and that a part of the fumes is frequently drawn off through the back wall of the stall and discharged into the air through a high chimney. A very convenient form of stall for this purpose is that shown in Figs. 33 and 34, which is in some places used for roasting copper ores.

At *Przibram* (Bohemia) stalls of more or less similar type are used for

\* *Trans. A.I.M.E.*, vol. xvi., p. 23.

roasting lead matte. They are built of slag-brick, and, besides the ordinary draught inwards through the loosely-built front walls, are provided with horizontal passages in the side walls, as shown in Figs. 33 and 34, which expedite the roasting and also bring it more under control. The well-roasted



Figs. 33 and 34.—Roasting-stalls.

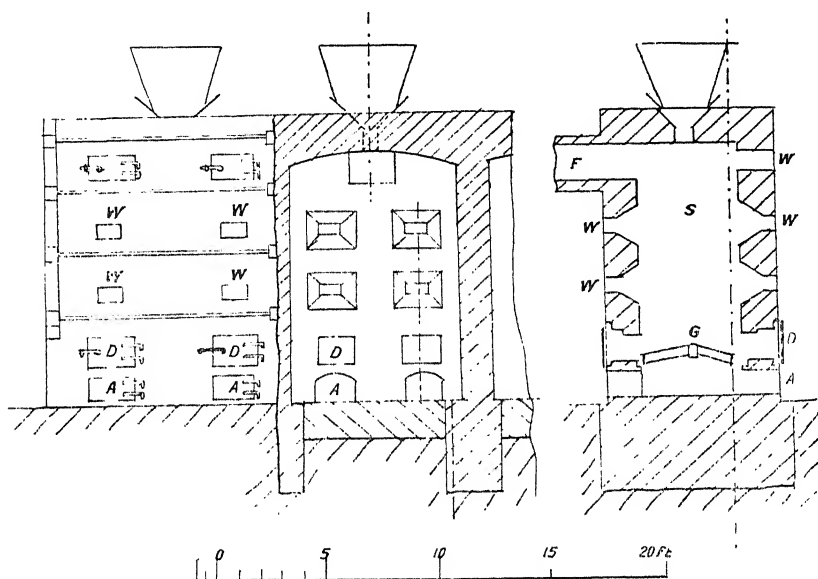
matte is about half the total charge of the stalls, and contains 8 to 10% sulphur; all lumps containing more are re-roasted. The consumption of fuel is, of course, less than in heaps.

Similar stalls are in use at *San Luis Potosi* (Mexico) and many other places for roasting leady mattes.

The *Wellner* stalls used at Freiberg have a sloping bottom, and require no "roast-bed" of wood; each half stall measures 10 to 12 feet wide by 8 to 10 feet deep from front to back.

The only advantage of stalls of this kind is that any kind of cheap inferior fuel can be used in them; they are, however, awkward and expensive to fill and empty, especially when wood of even inferior quality is scarce and dear. They cannot be used at all for high grade non-pyritic lead ores, but are still used for pyritic ores and for mattes at some out-of-the-way places, where the quantities handled are small and the country round is not in a high state of cultivation.

**Kiln-roasting.**—Kilns are furnaces of the general type of an ordinary pyrites burner, which, however, is unsuitable for leady ores, inasmuch as the column of ore is too low to give the required temperature.



Figs. 35 and 36.—Freiberg Kiln.

Kiln-roasting for pyritous lead ores in lump form is only adopted when the roast-gases produced can be usefully made into sulphuric acid, or when they must be rendered innocuous to the surrounding country. The ores treated must be poor in lead, or it is impossible to prevent them from sintering. No fuel is required in kiln-roasting, since, once started, the necessary heat is supplied by combustion of the sulphur; skilled labour is, however, indispensable, and the roasting with leady ores is so imperfect that it must be followed by a re-roasting of the lump ore in heaps, and of the smalls in reverberatory furnaces.

Lead ores containing much pyrites are successfully roasted in kilns at Oker (Harz) and at Freiberg. The construction of those at the latter place

is shown in Figs. 35 and 36. S is the shaft of the kiln,  $9\frac{1}{2}$  feet high,  $7\frac{1}{2}$  feet long, and 4 feet 3 inches wide. It is filled with ore through an opening in the arch by means of a hopper closed by a sliding door, or, better, by means of a cone and bell. The ore falls upon the pierced saddle-grate, G, and is ignited through the ashpit doors, A, through which also the supply of air is regulated. The roast-gases are led off through a flue, F, in the back wall into a main flue, which is common to the whole system of kilns, and communicates with the sulphuric acid works. The roasted ore is raked out through the discharging doors, D, and the descent of the column is regulated by means of the working doors, W, through which bars can be driven into the charge to loosen it and break up half-fused masses. Each kiln turns out  $1\frac{1}{2}$  tons of roasted ore per twenty-four hours.

Mattes are also roasted in these same kilns.

At *Oker* similar kilns to those at *Freiberg*, but somewhat higher, arranged in groups of five, all communicating with each other under the arch which forms the roof, are used for roasting mixed copper and lead ores. Such ores, averaging blende 25 per cent., pyrites 25 per cent., chalcopyrite 15 per cent., galena 14 per cent., heavy spar 11 per cent., and other gangue 7 per cent., and of which 1·8 to 2 tons are charged per twenty-four hours, cannot be sufficiently desulphurised at one roasting. They are, therefore, re-roasted twice in heaps to fit them for smelting.

At *Lautenthal* (Oberharz) kilns of the *Freiberg* type, arranged in groups of six, and larger than the *Freiberg* kilns, being 13 feet 10 inches high, 7 feet  $6\frac{1}{2}$  inches wide, and 8 feet 4 inches deep, are employed for roasting mattes. From the centre of each side rise saddle-shaped ridges dividing the roasted charge into four portions corresponding with the four doors, which facilitates the drawing. In order to prevent sintering, four parts of fresh matte are mixed with three parts of once roasted matte to form the charge, and the total output of each kiln is 2·2 tons per twenty-four hours, the roasted matte still, however, containing about 8 per cent. sulphur. The roast-gases contain by volume 5 to 6 per cent.  $\text{SO}_2$ , and the results obtained are better on this material than with the pyrites burners formerly in use, the product of which still contained 13 per cent. sulphur. These kilns are, of course, somewhat expensive to build, and their output is small, but the local demand for sulphuric acid in the neighbourhood renders the recovery of the sulphurous gases profitable.

At *Casapalca* (Peru)\* a complex pyritic zincy lead ore is roasted preparatory to smelting in *Guyer* kilns of somewhat similar design, 18 feet long, each of which roasts 10 tons per day from 30 per cent. S down to about 10 per cent., with a fuel consumption of about 1 per cent. of fine coke breeze, and the total cost of roasting does not exceed 1s. per ton with cheap labour.

**Comparison between Different Systems of Roasting Ore and Matte in Lump Form.**—The following table shows at a glance the advantages and disadvantages possessed by heaps, stalls, and kilns respectively :—

\* *Private communication, J. Woodgate.*

	Heaps.	Stalls.	Kilns.
Cost of erection and repairs, . . .	Nil.	Medium.	High.
Cost of working, labour, . . .	About equal in quantity.		
Cost of fuel, . . .	High.	Medium.	Almost nil.
Perfection of roast with ordinary leady ores, . . . }	Poor.	{ Better than heaps.	Worse than heaps.
Utilisation of roast-gases, . . .	Nil.	{ Possible in part. }	Almost complete.
Danger of loss of silver by dusting and leaching, . . . }	Greatest.	Intermediate.	Least.

It is, of course, understood that only the coarser parts of ores which are rich in pyrites can be at all advantageously roasted in lump form. Such ores, however, which are usually roasted in heaps, can frequently be handled to greater advantage in stalls or kilns. In many cases the value of the sulphuric acid produced by condensation of the roast-gases added to the amount of compensation for damage to crops, &c., saved by preventing the escape of the fumes will more than pay interest and wear and tear on a complete plant of kilns and acid chambers; again, in other cases, the arsenic contained in the ore will pay a good interest on the cost of kilns and settling flues, even if the whole of the sulphurous and sulphuric acids in the roast-gases be allowed to escape; while in cases where none of these bye-products can be advantageously utilised or rendered marketable, and when the surrounding country is barren, simple heap-roasting is often the cheapest and best system, even when it has to be repeated on part of the ore.

**Sintering of Slimes.\***—Lead slimes in rough air-dried lumps or in briquettes can be readily and cheaply heap-roasted in a dry climate. At the works of the *Broken Hill Proprietary Co.*, more than 100,000 tons of slime have been treated in this way. The slimes have the following general composition:—PbS 24 per cent., ZnS 29 per cent., FeS<sub>2</sub> 3·4 per cent., silica and silicates of manganese, alumina, iron, and lime (in rhodonite, garnets, and kaolinised felspar) 44 per cent., and they are run from the dressing works into large shallow pans about 9 to 12 inches deep, where they are allowed to dry for a couple of days before being cut up with spades into rough blocks 4 to 6 inches across. These are built into heaps about 20 to 22 feet wide  $\times$  6 feet 6 inches high and 200 to 250 feet long, holding 1,000 to 1,500 tons each, with brick fireplaces and brick air-channels at the bottom. About 1½ per cent. of wood in the fireplaces is required for kindling, an operation which lasts only about 24 hours, after which the heaps take 10 to 15 days to burn out, care in adding a covering of fresh fines being sometimes required to check too fierce local heat. The outside covering, which is only partially roasted, is removed, pugged with water, and used for plastering over a new heap. The inside is found to have sintered into a porous, but coherent, mass, which only requires to be broken up into pieces of manageable size. A comparison of the composition of the raw and roasted material is as follows:—

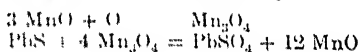
	Pb per cent.	Zn per cent.	Sulphur per cent.	Oz. Silver.
Raw slime, . . . . .	17·0	16·0	12·5	17·5
Sintered slime, . . . . .	14·5	12·5	7·1	15·8

\* Horwood, *Trans. Aust. I.M.E.*, vol. ix., pt. 1; and Delprat, *E. and M. J.*, Feb. 16, 1907, p. 321.

The residual sulphur is mostly present as sulphate; the loss of weight in the process is 5 per cent., a good deal of which, however, is no doubt lost by dusting.

The process is cheap, and the product is in admirable condition for the smelting furnaces; the losses of lead and silver are, however, considerable, amounting to  $12\frac{1}{2}$  per cent. of the former and 6 per cent. of the latter. As the expulsion of sulphur is also imperfect, the process must be looked upon from the purely technical point of view as crude and wasteful. The metallic contents of Broken Hill ores being, however, very great, and lead and silver being the cheapest things available, while fuel and labour are the dearest, from the commercial point of view the process is highly successful, no other equally cheap method of roasting these slimes having been devised.

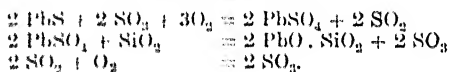
The reactions which take place in this sinter-roasting of lead slimes are somewhat obscure. It is clear that at a temperature of  $400^{\circ}$  C. the PbS is converted into basic sulphate, and that at  $800^{\circ}$  C. this is decomposed with formation of basic silicate. It has been suggested that the CaO and MnO virtually or potentially liberated from their silicates through the action upon them at a high temperature of the first PbO formed, may themselves become oxygen-carriers, according to the following equation:—



and also



The extreme porosity of the material, however, would alone be sufficient to account for the rapidity of the roasting process, which, in the author's opinion, is also largely assisted by the following reactions:—



#### ROASTING FINE ORES AND CRUSHED MATTES.

The greater part of the lead ore received by any smelting establishment is in a more or less fine condition, whether in the form of natural smalls from the mines or, as is more common, of concentrates from dressing works, which may vary from bean size jig concentrates down to the finest dust from revolving tables and other slime machines. The variety of appliances which have been used for roasting fine pyritic ores is great; they may be classified as reverberatory, cylinder, muffle, and shaft furnaces, examples of all of which will be found described in other volumes of this series. The roasting of lead ores presents, however, much more difficulty than that of copper ores, since they are so much more fusible, while it is often complicated by the presence of large amounts of blende, one of the most difficult of all metallic sulphides to thoroughly oxidise. Some of the roasting furnaces which have been most successful on pyritic copper ores have proved quite unsuitable to the roasting of ordinary galena ores carrying blende; practically speaking, nearly all lead ores are roasted in some variety of reverberatory furnace, either hand stirred or mechanical.

As regards mattes, these are obtained naturally in lump form, and, if roasting in heaps, stalls, and kilns were not so imperfect, one of those methods would be universally adopted. Matte, however, is usually more intrinsically valuable than the average ore put into the smelting furnaces, so that the losses in handling and in treatment become correspondingly more important; it is, at the same time, much more difficult to roast in a lump form, owing to its want of porosity and to its ready fusibility; in view of its further treatment, moreover, it is essential that the roast should be as perfect as possible. For all these concurrent reasons, the practice of crushing matte through a 4- to 8-mesh screen preparatory to roasting has become almost invariable in modern works of any size. Lead ores containing upwards of 15 per cent. of that metal should be crushed through a 8- to 12-mesh for really good roasting.

In all cases when leady ores and mattes are roasted in furnaces, a system of flues and condensing chambers must be provided in order to catch, not only the particles of ore mechanically carried over by the draught, but also the lead which is volatilised in the metallic condition through the reaction of  $\text{PbO}$  and  $\text{PbSO}_4$  on  $\text{PbS}$  in the hottest part of the furnace, and the still larger amount of  $\text{PbS}$  there volatilised\*—both of which, of course, become oxidised subsequently to vaporisation.

#### REVERBERATORY FURNACES.

The original style of reverberatory furnace in which the charge was spread evenly for roasting, and from which it was raked at the end of each operation, is now probably extinct, as it is not only of small capacity, but is extremely wasteful of fuel.

The type of furnace which has been most uniformly successful in roasting lead ores is the old-fashioned long hearth reverberatory, which, though not economical of fuel or of labour, is adapted to ores carrying any proportions of galena, pyrites, and blende. In order to roast any ore in a long hearth reverberatory, it is introduced at the cold end farthest from the fire, and gradually moved down towards the hottest part of the hearth, where it is discharged. The speed with which the material is moved forward depends upon its general composition, highly pyritic material can be roasted at a high temperature and passed through quickly; when galena predominates, however, the roast must be slow and the temperature kept low throughout; a high proportion of blende again means exposure for a length of time to a high temperature.

**Double Hearth Reverberatories** having two hearths superposed ("double-bedded calciners"), in order to save loss of heat by radiation, are still to be found at a few works in Germany and in other parts of Europe, but have been generally abandoned elsewhere on account of the heavy labour required to work the charge on the upper hearth by means of movable platforms on which the men stand, and of the difficulty of getting at the lower hearth and arch for effecting repairs when necessary. The saving in fuel effected by the superposition of hearths, though real enough, is, in most cases, insufficient to compensate for the above

\* Lead sulphide being volatile at a lower temperature than the metal itself, its oxide or sulphate.

disadvantages. Beyond this and a slight saving in first cost, the chief advantage of the "double-bedded calciner" is the economy of space in old metallurgical works where there is not room to erect new furnaces of the long type.

The superposed hearths of the Meckernich furnaces are very long, being each 50 feet long by 13 feet wide. In twenty-four hours, each furnace puts through 9 tons of lead ore dressed up to 50 to 60 per cent. lead, with a consumption of 27 cwts. of coal (*i.e.*, 15 per cent. by weight of the ore), the roasted ore, which is sintered together before being drawn, averaging only 0.6 to 0.7 per cent. of sulphur. No such perfect roasting of lead ore is conducted in America, but it takes a good deal of labour, and would be, in most cases, far too expensive under American conditions.

**Single Long-Hearth Reverberatories.**—As has been already stated, the type of roasting furnace in most common use is the single long hearth reverberatory, called by the Germans "*Fortschaufelungsofen*," in which are to be found at the same time several charges in different stages of the process, each of which is successively raked and pushed forward as a completed charge is drawn.

When the subsequent treatment of the roasted ore requires it to be in a fused or agglomerated condition, the roasting hearth at the firebox end terminates, either in a "sump" or depression in which the semi-fused mass can be collected, or, preferably, in a vertical channel or flue through which the roasted ore is raked into a separate smelting hearth or fuse-box, and through which the flames passing over the latter ascend to the roasting hearth. When a separate fuse-box is used, the ore is almost always made so fluid as to be drawn off through a spout into slag pots, and only if very infusible is it raked out in agglomerated cakes and masses through the working doors. In America, when agglomeration of the ore is desired, this separate smelting hearth is almost invariably employed, while in Europe, on the other hand, the continuous hearth with sump is in general use.

The disadvantage of the latter is that the heat cannot be so well concentrated, and the mass begins to soften and sinter together as far back as the third or fourth door from the fire-bridge; this not only increases the difficulty of rabbling, but, unless the hearth is exceedingly long, results in an imperfect roast, particles of unaltered sulphides becoming glazed over, and so protected from oxidising influences. On the other hand, the volatilisation losses both of lead and of the precious metals are much less in the sump reverberatory than in the separate fusion hearth. The arguments for and against "sintering" and "slagging" respectively have been already stated in this chapter, and need not be repeated here. Suffice it to say that the rich non ferruginous ores commonly treated in Europe are best roasted in the sump form of reverberatory, while the ferruginous concentrates and ores which are more usually met with in America were, prior to the advent of blast roasting, best treated in that with separate fusion hearth.

**General Considerations.** The length of the hearth from flue to firebridge, or to the beginning of the smelting hearth, as the case may be, varies from about 40 feet, in the case of galena ores comparatively free from pyrites, up to 60 or 70 feet, in the case of ores containing much pyrites, half of the sulphur in which is available as fuel. Reverberatory hearths have been constructed of much greater length than these, but it may be safely stated that



no increase above the maximum length of about 70 feet is warranted, for any further trifling economy of fuel is more than offset by the increased labour required to handle the charges, the increased cost of construction resulting in no compensating advantage. The width of the hearth should be as great as possible, consistent with reasonable facility in handling the ore. It may be as little as 9 feet 6 inches and as much as 17 feet, but the capacity of the narrow hearths is much smaller in proportion to their cost, and modern furnaces are generally from 14 to 16 feet wide. Beyond 17 feet nothing would be gained by increasing the width, for, in the first place, the handling of the charge would be more difficult, and, in the second place, arches of much more than that span at the very low rise of  $\frac{1}{2}$  to  $\frac{3}{4}$  inch to the foot generally adopted would not be safe under the alternate expansion and contraction caused by changes of temperature.

It is of great importance to make the arch as flat as possible, so that the flames may be well spread out towards the sides, instead of travelling along in the central channel of the roof, and doing comparatively little work upon the ore. For the more complete utilisation of the heat contained in the products of combustion before their escape from the furnace, the arch is sometimes built with a slight downward inclination toward the flue. The same effect, however, is much more readily produced by inclining the hearth upwards in the same direction, either by giving it a uniform gentle slope or in steps, the latter plan being preferred by some on account of marking off conveniently the area occupied by each of the separate charges.

The arch is best constructed of firebrick throughout,\* though it is quite usual to employ selected red brick in the part furthest from the fire. The hearth is always constructed of the best firebrick obtainable, so as to better resist the wear and tear of the tools. At the far end of the furnace the bricks are laid on edge (and sometimes even on the side, though this is really poor economy), but near the firebridge they are best laid on end, and, of course, the sump or fusion hearth must be so laid.

The number of working doors is made as small as possible, so as to reduce both the first cost and the subsequent cooling of the furnace; but if they are placed too far apart the difficulty of handling the charges is much increased. In practice the distance varies between 6 and 8 feet, the most common distance being from 7 feet to 7 feet 3 inches from centre to centre of the cast-iron door frames. The blocks of masonry between adjacent working doors are in modern furnaces kept as small as possible, in order to facilitate access to all parts of the hearth. The skew-backs of the arch must be supported between the door frames, either by means of special plates cast with a strengthening rib; of wrought-iron I girders inserted behind the buck-staves; or, finally, of a sheet of boiler plate, say 18 inches wide by 1 inch thick. The buck-staves are frequently made of steel rails, which in some places are more convenient to obtain than I beams, but the use of the latter is preferable, since much greater strength can be got out of a smaller weight of metal than by the use of rails.

An essential condition of quick and economical roasting, often lost sight of, is the provision of a supply of heated air in close proximity to the surface of the ore. Merely passing an excess of air through the fireboxes, besides being wasteful of fuel, is not nearly so effective as furnishing a separate

\* Unless they cost over three times the price of common brick.

supply of fresh air previously heated by passage through the hollow division walls of the firebox, through the bridge, or even through hollow flue walls, and under the hearth. This is particularly advisable in the case of pyritic ores and of mattes.

The thickness of the layer of ore upon the hearth may vary between 3 and 6 inches, this depending partly on the local cost of labour, but more upon the nature of the ore. Pyritic material, which readily parts with its sulphur, can be roasted well in 5 to 6-inch layers, whereas ores rich in lead and zinc, as well as mattes, require a thin layer and much rabbling in order to expose fresh surfaces, and to obviate the tendency to "crusting" or incipient sintering, which shows itself more the thicker the layer upon the hearth and the higher the percentage of lead.

The fuel used in roasting furnaces may be of the cheapest description available, provided that it makes a certain amount of flame. Wood is frequently employed where it can be obtained cheaply, but slack coal is much more commonly used. Oil and producer gas have been used in a few instances, and the use of both appears to be spreading. At *Selby* (Cal.)\* crude oil at 7s. per barrel is used in four long Ropp furnaces by means of eleven burners,† steam being employed as atomiser, and shows an economy in fuel cost of 40 to 60 per cent. over coal at 21s. 8d. per short ton; besides the saving in cost there is a certain increase in roasting capacity, owing to the smaller amount of products of combustion, and to the better oxidising atmosphere which can be maintained with regularity, on account of the perfect control over the burners.

The quantity of ore put through in twenty-four hours depends on the percentage of sulphur expelled; on the prevailing mineral accompanying the galena, whether pyrites, quartz, or blende, the first being the quickest and the last the slowest to roast; on the efficiency of the workmen; and on the final condition of the roasted product, whether pulverulent, sintered, or fused. The capacity of furnaces of the same area may vary in accordance with the above factors between 6 and 14 tons per day, the average lying between 8 and 12 tons. The consumption of fuel is greater than with pyritic copper ores; if coal be used it varies between 15 and 30 per cent. of the weight of ore put through, averaging 20 to 25 per cent., according to the quality. If wood be used, the consumption may vary between two and four cords per day, as it may be reckoned that two cords of wood will be a little more than equal to 1 ton of coal. The amount of labour required varies also with the composition of the ore and the condition of the roasted product, but, as a rule, three men are required to a shift, which may be either of eight or of twelve hours, generally the former in America and Australia, the latter in Europe. If the roasted ore is turned out in a powdery condition, a capacity of 2 tons per man may be regarded as a reasonable maximum; if it is to be slagged, the capacity is rarely, if ever, over  $1\frac{1}{2}$  tons per man, and may fall below 1 ton per man.

**Examples of Reverberatory Furnaces.**—As an example of the long hearth reverberatory arranged for plain roasting without sintering, that in use at *Przibram* may be quoted, and is shown in Figs. 37, 38, and 39.‡ In

\* A. v. d. Ropp, *Mining and Scientific Press*, Nov. 29, 1902.

† For particulars of crude oil burners, see Chap. xvi., Fig. 266.

‡ From Schnabel, *Handbuch der Metallhüttenkunde*, vol. i., p. 334.

these F represents the fireplace fired at both ends; *b*, the bridge, built of magnesia brick and cooled by a long cast-iron water-box; H is the hearth, 48 feet long and 10 feet 3 inches in average width, or equal to 12 feet 4 inches as usually measured. It will be noticed that the fourteen working doors, W, are not placed in pairs opposite each other as in the Freiberg and most American furnaces, but alternating; this arrangement much facilitates the even working of the charge, and is widely adopted in the most modern roasters. The ore is let down through the charging pipe, *c*, at the flue end of the hearth in charges of 1 ton each, and spread out through the two doors at that end, being gradually worked forward towards the bridge, and finally, when roasted, raked through the discharging doors, *d*, into a small truck running in the

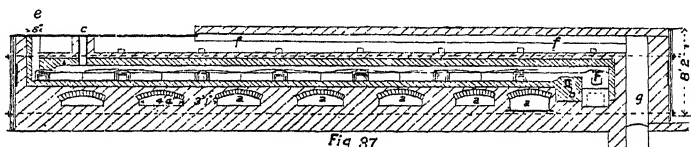


Fig. 37.

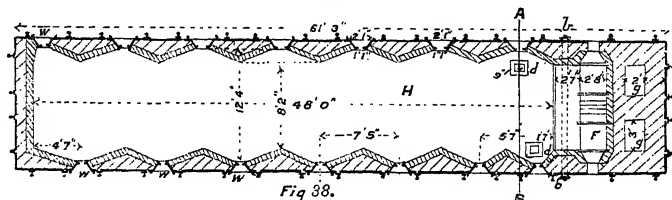
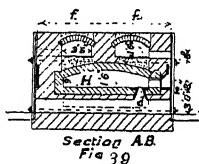
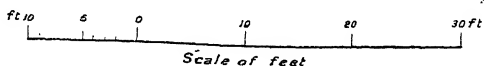


Fig. 38.

Section A-B  
Fig. 39.

Figs. 37 to 39.—Reverberatory Furnace for Plain Roasting (Przibram).

first arch, *a*, which is provided with a floor of cast-iron plates. Similar arches run across the furnace at intervals, their function being merely to cool the hearth. The furnace gases rise through the vertical flue, *e*, and turn back under a floor of cast-iron plates which serves for drying the ore, after which they follow two brick flues, *f*, built over the firebrick arch and fire grate, finally turning down beyond the latter through the flues, *g*, which are provided with dampers and connected with a main underground flue.

The ore roasted in this furnace contains 51 to 58 per cent. galena and 8 to 11 per cent. blende, with 12 to 14 per cent. silica, the remainder being chiefly iron pyrites and chalybite. It is roasted down to an average of only 1 per cent. S, and is very slightly agglomerated with the rabbles at the end of the operation before dropping into the hot-ore truck.

The Freiberg furnace for sinter roasting is shown in Figs. 40 to 42. It is somewhat similar to that just described, but the portion of the hearth

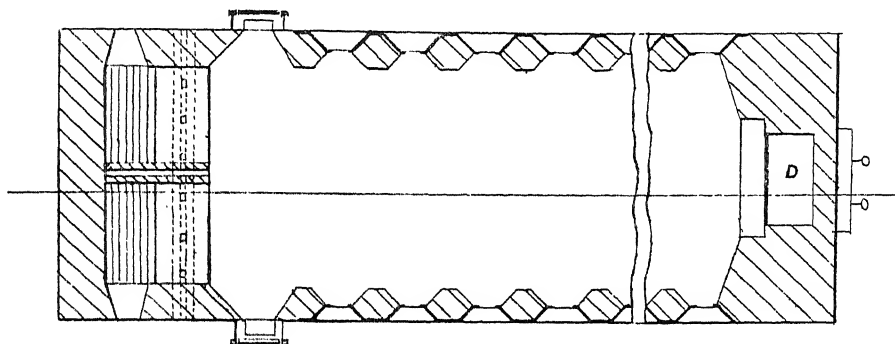


Fig. 40.

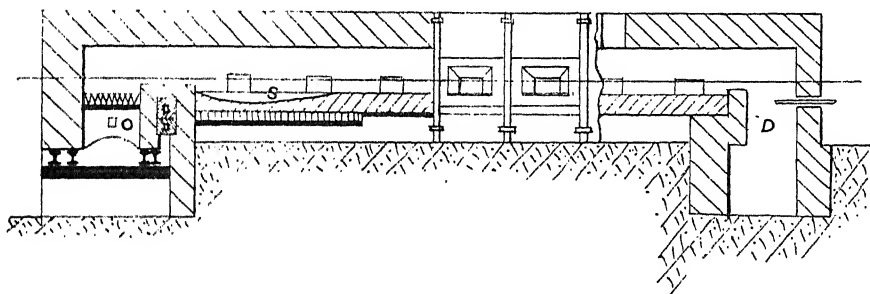


Fig. 41.

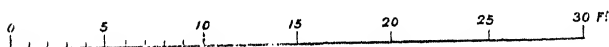
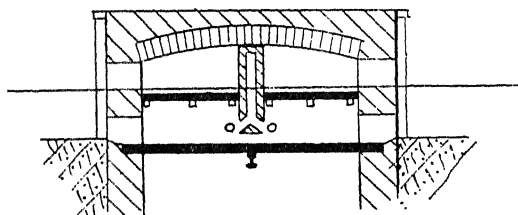


Fig. 42.

Figs. 40 to 42.—Reverberatory Furnace for Sinter Roasting (Freiberg).

marked S is the sump, sloping gradually to a trough between the first working doors, through which the agglomerated ore is raked into iron wheelbarrows.

With very rich lead ores the melting point is so low that perfect *fusion* of the charge can be easily effected in a sump furnace without separate fuse-box. This was done at *Mine La Motte* (Mo.)\* in the case of a rich galena concentrate poor in silver. The hearths vary in length from 55 to 66 feet, and in width from  $11\frac{1}{2}$  to 13 feet, the length being generally five times the width, with ten doors on each side, and a continuous slope of 18 to 20 inches. The grate is made 3 feet 6 inches deep from bridge to grate bars, so as to use wood fuel. The bridge and firebox division wall are air-cooled, a small chimney on the roof above the latter drawing a strong current of air through both ends of the hollow bridge and keeping it cool. As in the Freiberg furnace, the "sump" in which the ore is fused down is composed of a single course of firebrick built into a wrought-iron pan resting on brick pillars, between which air circulates freely; this air cooling is very effective in preventing corrosion.

The dressed galena contains 65 to 75 per cent. Pb, and the "sulphides" (a sort of middle product of galena and pyrites) 15 to 30 per cent. Pb, 16 to 45 per cent. Fe, and 1 to 5 per cent. Ni and Co. The roasting charges are made up of a mixture of both products, 8 tons per day being treated in the 55 by  $11\frac{1}{2}$  feet furnaces, and 10 tons in the 65 by 13 feet furnace. The roasting is carried on in the following way:—†

As soon as one fused charge is drawn, sand is thrown over the sintering hearth near the bridge and the fire lowered while the next charge (already desulphurised down to 5 or 6 per cent. sulphur) is spread over it, and each successive charge moved forward two doors, a fresh charge being dropped in at the flue end of the furnace and spread. This operation of moving forward and adding a new charge takes about two hours. The doors are now closed and the fires cleaned and fed; the proper amount of sand is thrown upon the charge to be fused and well rabbled in. The fire is then urged for four hours, the fuse charge being well stirred every hour. When ready for drawing, the men, protected by a half-cylinder of boiler plate, which stands on the ground below and in front of the working door, rabble out the fused mass (which is described as of a "ropy" consistency) with long hoes direct through the working doors, and subsequently break it up by throwing water on it, when it cracks and is easily shovelled into wheelbarrows. Sometimes a little lead is reduced in the furnace by "reactions," but this trickles away through a small hole in the bridge into the ashpit, whence it is recovered by washing the ashes. Eight tons of ore are roasted and slagged per twenty-four hours. Other data are given in Table XI.

The "*Colorado*" roaster, with separate "fuse-box" or smelting hearth, is shown in Figs. 43 to 47. The particular furnace figured is from the *El Paso Smelting Works*; it is peculiar in that the furnace gases are turned under the hearth through vaults occupying the whole width, which thus become dust chambers. The arrangement is a poor one, as the cleaning out of dust chambers in such a situation is a particularly hot and dusty job, and more or less interferes with all the work of the roaster shed while it is going on; it is only defensible where there is no room for proper dust chambers. Vaults under the hearth are never required, and the space is much better filled with rubble masonry or simply with well-tamped sand, as the temperature of the

\* Hofman, *Metallurgy of Lead*, 1893, p. 166.

† Kemp, *S. of M. Quarterly*, vol. x., No. 3, p. 215.

Fig. 43.

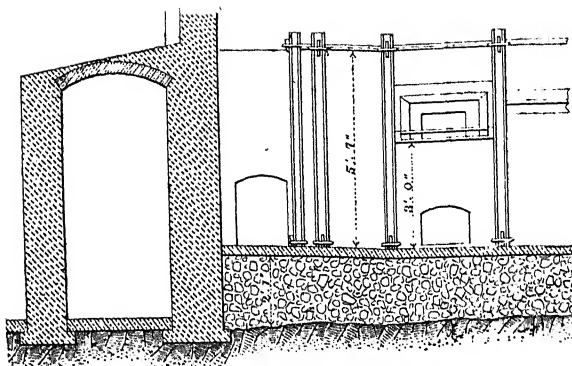


Fig. 44.

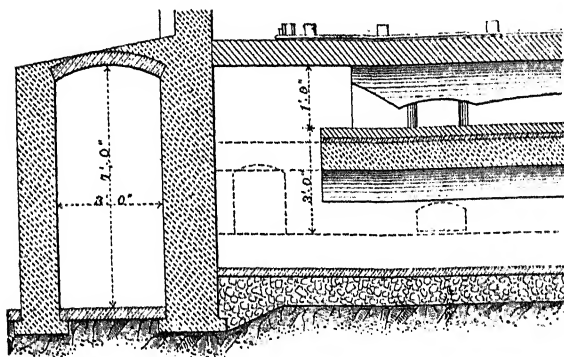
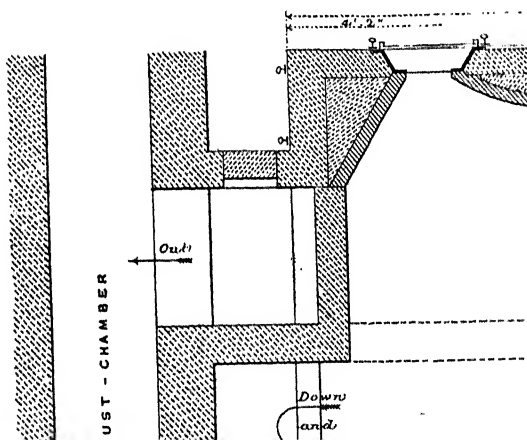


Fig. 45.



Plant	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.	12.	13.
Plum Creek, Tenn.	A	A	A	A	B	B	D	A	C	A	C	A	A
Type of furnace	Plain	Plain	Plain	Plain	Sinter	Sinter	Sinter	Sinter	Slag	Plain	Slag	Plain	Plain
Kind of roasting performed	...	...	...	...	...	...	...	...	...	...	...	...	...
Length of hearth, feet	48 0	30 0	40 0	62 6	48 9	25 6	49 2	65 0	69 0	...	40 0	...	60' 0"
Breadth of " "	10 3	12 0	17 0	12 6	19 8	11 6	13 1	15 6	15 6	...	11 6	...	16' 0"
Area of " " in square feet	466	468	680	759	459	682	1286	975	903	...	440	...	960
Area of grate	22 6	...	...	...	...	36	...	40	42	...	25	...	4200
Weight of each charge, lbs.	2265	3900	2750	2265	3897	4000	...	3600	...	...	...	4000	6
Number of charges per twenty-four hours	4	4	9	10	8	4	...	8	...	...	...	6	25,200
Total output, tons, per twenty-four hours	8820	13,200	24,846	22,050	26,460	16,000	19,845	28,500	22,000	...	10,000	24,000	25,200
Quantity in furnace at once, tons	15,485	13,200	8280	11,025	16,585	32,000	...	36,000	...	...	23	24,000	26
Output in lbs. per square foot of area	31	28	12	29	36	51	...	37	...	...	30	...	24
Time to roast each charge, hours	42	24	8	12	15	48	...	30	...	...	15	...	22
Percentage of heat in raw charge	35-40	50	35-50	25-30	20-30	50-65	50-60	25-39	...	...	4	...	22
" sulphur in raw charge	12	24	20	14	20	16-20	0-6-0-7	4-5	8	...	...	3	4
" in roasted charge	0-0	6-7	8-10	6-7	3-4	4-7	...	...	...	...	...	Gas	Coal
Description of fuel used	Lignite	Coal	Coal	Coal	Coal	Wood	Coal	Coal	Coal	Coal	Coal	Coal	Coal
Percentage consumption of nitro	35-38	18-20	14	20	25	62	15	24	27	26	...	23	33
Cost of fuel used per ton	...	20	4	3 3	5 4	...	...	6 3	5 9	6 3	...	2/2	...
" roasting labour	...	5	2 10	1/10	1/8	...	...	1/7	2	2	...	2/2	...
" fuel	...	4	...	...	...	...	...	...	1 6	3 2	...	...	...
Tools, repairs, and depreciation	...	1	2 6	...	1/	...	...	...	...	...	...	...	...
Total cost per ton of ore	...	9 4	8 4	6/ -	8/	...	...	5 11 1/2	9/3	11/5	...	5/	...

*Types of fuses.*—A is plain reverberatory; B, reverberatory with separate fuse-box; C, reverberatory with sump; D, double furnace; E, reverberatory with sump at end of lower hearth.

superposed heart reverberatory having sum at end of lower neurite.  
*References*.—1 and 5. Schnabel, *Handbuch der Metallatankunst*, vol. i., p. 335. 2. *Private notes*, 1904. 3 and 4. *Private notes*, 1897. 6. *Kemp*, S. of M.Q. vol. x., p. 215. 7. S., 9, and 10. *Private notes*, 1888, 1897, 1902, and 1902 respectively. 11. Brinsmade, *Miner. and Metall.*, 1902, xxii., p. 300, and abstract in *Min. Ind.*, vol. x., 1903, p. 437. 12. *Private notes*, 1902. 13. *Min. Ind.*, vol. xvi., p. 665.

calcining hearth rarely gets above a dull red, except very near the bridge. The fusion hearth must, of course, be built on a vault.

The furnace figured has four hearths, each 14 feet square, separated by means of 3-inch steps, the end farthest from the fire being thus closest to the roof, so as better to utilise the heat of the roast gases. The same result might be attained by dispensing with the drops and giving the hearth a continuous gentle downward slope towards the bridge.

The fusion hearth is nearly circular in plan, the radius being 10 feet 8 inches; connection is made with it from the smelting hearth by means of a vertical flue 26 inches high and 5 feet 6 inches long by 16 inches wide, the object being to ensure a sharp separation between the smelting temperature below and a roasting heat above where the hot gases suddenly spread out from 5 feet 6 inches to the full width of 14 feet. In this way the ore above is to a considerable extent prevented from fritting together until the moment of its arrival on the fusion hearth, a result favourable to the handling of the charge as well as to the perfection of the roast, and impossible to obtain with the Freiberg form of furnace, in which, with fusible ores, the charge sometimes begins to agglomerate as far back as the third or fourth pair of doors.

The arch of the furnace is generally built of firebrick only up as far as the end of the lowest roasting hearth, the rest being picked red brick, but some metallurgists prefer to build the whole arch of firebrick, as being more lasting and avoiding the evils caused by unequal expansion and contraction. The bottom of the roasting hearth is formed of firebrick set edgewise as closely as possible on a bed of red brick; while the working bottom of the fuse-box, formerly built up (as shown in figures) of quartz sand melted down with a small quantity of slag on a concave brick bottom, is now commonly made of a simple inverted arch of firebrick properly wedged up, which is found to be very durable and satisfactory. The whole of the fusion hearth is enclosed in heavy cast-iron plates, and a heavy plate is also inserted in the air-cooled firebridge to take the longitudinal stress off the hearth. The walls of the smelting hearth and the flue leading to it from the roasting hearth wear rapidly, and have to be patched with fireclay from time to time. The cost of a furnace like that figured is, at Denver, about £600, on the Mexican frontier about £1,000. Each furnace requires 86,000 red bricks, 15,000 firebricks, 10,000 lbs. of old rails, 12,000 lbs. of cast iron, and 4,400 lbs. of wrought iron (sheet iron and tie-rods).

The weight of charge is from 2,400 to 3,300 lbs.; it is dropped through a hopper at the far end of the furnace and spread evenly over the space between the last four working doors, remaining there till a charge is again drawn from the fuse-box, and the other charges have been moved forward a step, when it is all turned over and transferred to the second hearth. Besides the moving forward, each charge receives from one to three rakings with the rabble in the intervals, so as to expose new surfaces. When it reaches the end of the roasting hearth, and before it is dropped on to the fusion hearth, the latter is prepared by spreading on it a few cwts. of siliceous ore to protect the bottom, and to form slag with the ferruginous roasted ore. Sometimes, also, flue dust is prepared for subsequent furnace treatment by being charged into the fusion hearth, where it is soon melted down after being covered with the hot-roasted ore. As soon as the roasted charge has



been spread the fire is urged as much as possible; fusion soon commences near the top, but frequent and vigorous rabblings at short intervals are required to lift up the siliceous ore and bring it into intimate contact with the more basic material and slag already formed. After a few hours' hot firing the whole mass becomes fluid, when it is rabbled out into ordinary slag pots and left to cool; after which, it is broken up into lumps the size of a cocoa-nut for the blast furnace. When the charge runs high in lead the portions first fused must be rabbled out at once, as otherwise they would act too much upon the hearth and walls.

Furnaces like that above described will roast and slag from 6 to 10 tons per day with a consumption of 30 to 45 per cent. of coal, and leave from 2 to 6 per cent. sulphur in the product. When plain roasting, two men are found sufficient on a 12-hour shift, but when the charge is fused a third man is required to attend to the fuse-box and the firing. With labour at 9s. 6d. to 10s. 6d. for twelve hours' work, and coal at 7s. 6d. per ton, the cost of roasting and fusing ordinary ores should not exceed 8s. per ton—though with unskilled men and slipshod superintendence it is frequently much higher.

At most smelting works of the present day the "fuse-box" has been done away with; in fact, the author is not aware of a case where it survives. Occasionally, however, where agglomeration in the furnace is practised, the place of the old fusion hearth is taken by a "sintering hearth," running the full width of the hearth, but at a lower level, the object being to make a sharp line between ore which is still roasting and that which is being slightly agglomerated prior to drawing. In some cases the slightly sintered ore is merely drawn into iron trucks, where it only partially coheres; in other cases it is rammed into ordinary slag pots by the furnacemen, where it forms masses nearly as coherent and, at the same time, as porous as coke, and, therefore, in the best possible condition for further treatment. The obtaining of a thoroughly coherent mass is in some unexplained way much facilitated by punching three or four holes in the mass immediately after ramming down, through which a copious discharge of fumes takes place.

*Other Examples of Long Reverberatories.*—At the *Cockle Creek Works* of the Sulphide Corporation, Limited, zinc-lead concentrates (middlings), and slimes containing 28 to 30 per cent. Pb and 25 to 28 per cent. Zn, were until recently roasted in long hand reverberatories, 62½ feet by 12 feet, at first, for leaching by the Ashcroft process (*v. Chap. xiii.*), and subsequently for direct smelting. A moderately high temperature with low draught is employed, and the roasted product contains about 8 to 10 per cent. SO<sub>2</sub>, besides some 3 per cent. of sulphur as unoxidised sulphide. Some further particulars are given in Table XI. This table contains similar particulars with reference to all the furnaces described, and to a great many others as well.

At various works, notably at the *Smelter No. 1* (Monterey) gas-producers have been used for firing roasters, with the double object of facilitating regulation of the temperature where the available labour is unskilled, and of utilising coke breeze which tends to accumulate around smelting plants. At the plant referred to there are twelve long hand-rabbed reverberatories fired by six Taylor gas-producers, each 7 feet in diameter by 16 feet high, the fuel used being a mixture of Laredo coal and coke breeze from which the fine dirt has been separated out.\*

\* *Private Notes*, 1902.

**Mechanical Furnaces on the Reverberatory Principle.**—Under this head might be considered the cylinder furnaces of various kinds, but the term “mechanical reverberatories” is usually restricted to those furnaces with a fixed bed in which the ore is rabbled or moved along by stirrers actuated by suitable machinery, and to those having a revolving circular bed, upon which the ore is moved by fixed stirrers. Of the former class are the O’Hara, the Brown-Allen and Brown Horse-shoe, the Pearce Turret, the Spence, Keller, Wethey, and Ropp furnaces. Of the latter class are the Brunton, Heberlein, and Godfrey furnaces.

In order to satisfactorily expel the greater part of the sulphur contents of leady ores without bringing about premature agglomeration, it is necessary to be able to accelerate or retard at will the progress of the ore from the coolest to the hottest part of the furnace, and to vary the amount of stirring at different stages of the roasting in accordance with the composition of the particular ore mixture being handled at the moment; since both the temperature required for oxidation of the sulphur and that required for fusion of the ore vary within wide limits, according to the predominance of galena, blende, or pyrites as the principal sulphide mineral present. The hand-rabbled reverberatory consequently can be readily adapted to treating a variety of ores and a varying ore mixture; mechanical furnaces, on the other hand, in which the motion of the rabbles, and consequently the motion of the ore through the hearth, are both uniform, are at a distinct disadvantage as regards the quality of the work done, and require for good results that the ore mixture supplied to them shall be of a nearly uniform composition, and comparatively low lead contents.

One of the greatest difficulties with roasting leady material in a mechanical furnace is the tendency to form crusts through incipient fusion in the lower part of the charge. In hand-worked furnaces the tools, of course, bear directly on the hearth, and so crusts can be broken up as soon as they begin to form while still soft; but it is impossible to let mechanical rabbles work actually dragging on the hearth on account of the friction, with the result that crusts grow from the bottom upwards, till they meet the rabble blades, and go on growing as these wear away. About twice a year, therefore, it is necessary to close down the furnace and cut up and plough up the crusts which have formed in the six months’ campaign, and, if left undisturbed, go on growing until the rabbles wear away. At some works a special curved steel plough is affixed to a blank rabble arm, along which it is made to move between each circuit, so as eventually to cover the whole hearth in successive portions, and so rip up all the crust in about an hour or two; this process is repeated once a week or even oftener. At other works the crust is allowed to grow freely until the rabbles wear out and have to be replaced, which takes place about twice a year, at which time the furnace is shut down, and the whole of the crusts are cut out by hand before re-starting.

**The Brown-Allen-O’Hara Furnace,\*** so largely used in the Western States of America for roasting pyritic copper ores and mattes, consists normally of two superposed reverberatory hearths 90 feet long by 8 feet wide in the clear. The ore is moved forward by a series of stirrer carriages, which run on an iron track separated from the roasting hearth by means of a slotted

\* For figures and details see Peters, *Modern Copper Smelting*, 7th edition, p. 202.

wall, and which are attached to each other by chains running over a system of vertical pulleys at each end of the hearth. The ore fed in automatically at one end of the upper hearth is gradually moved forward by the stirrers till it reaches the far end, when it drops through on to the lower hearth. The stirrers run on in the air, supported by the chains, for a certain distance in order to cool them; and, being turned over by the pulleys, they then return on the lower hearth and gradually work back the ore to the discharging

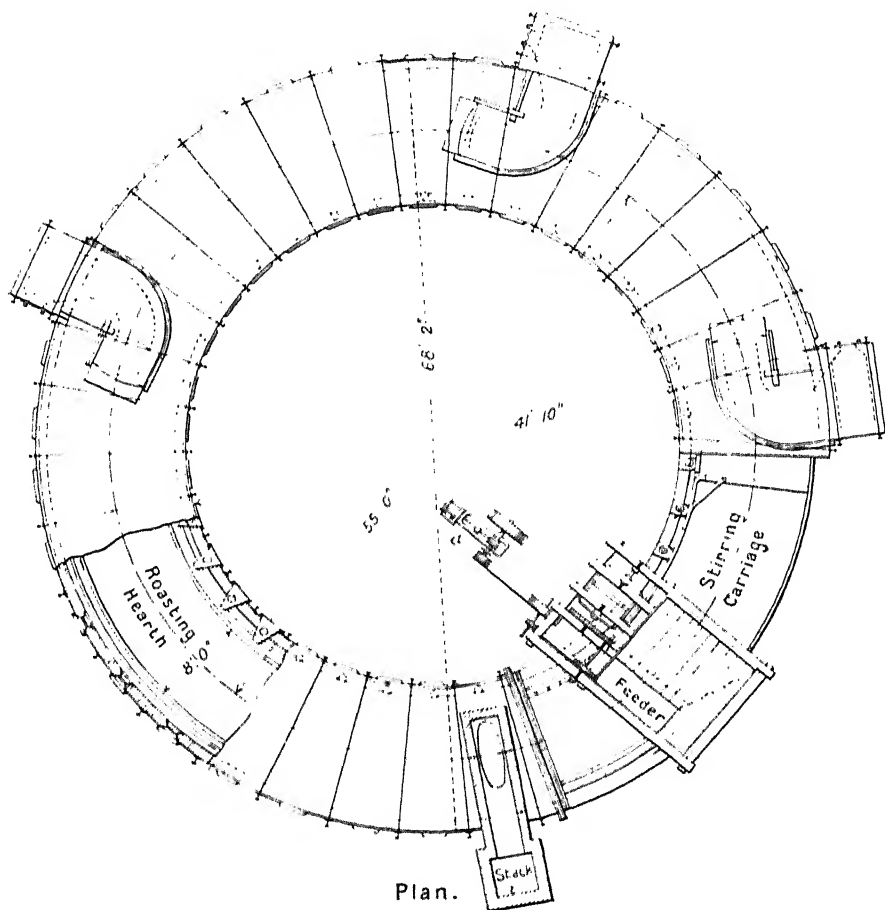


Fig. 48. — Brown Horse-shoe Furnace.

doors at the feed end of the furnace. The fireboxes, three or four in number, are built outside of the main furnace walls, the flames entering by cross arches.

These furnaces cost about £2,000 apiece in Denver or Pueblo, and their capacity on pyritic material is from 30 to 35 tons per day, according to the perfection of the roast, but the fuel consumption is high. They have been



used at the *Globe* (Denver), *Pueblo*, *Trail* (B.C.), and other lead-smelting plants, but owing to the formation of crusts they do not give satisfactory results on materials containing more than 15 to 20 per cent. lead as a maximum. Table XII. gives a few particulars about the performance of the furnace at the *Pueblo* plant, where the charge is frequently a mixture of finely crushed matte and galena.

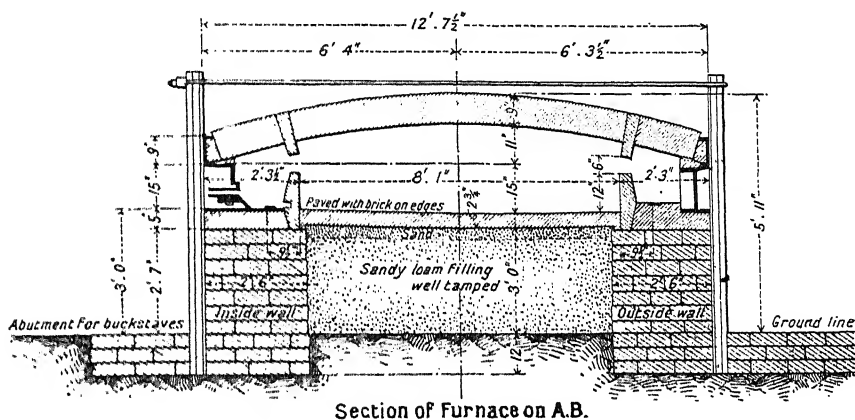


Fig. 49.

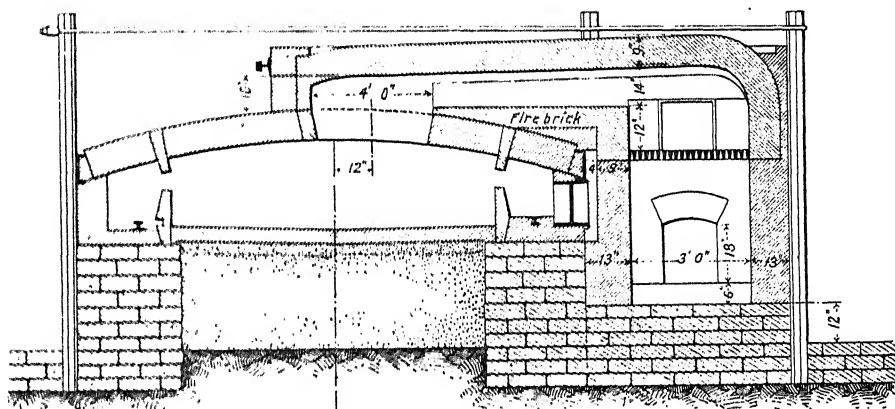


Fig. 50.

Figs. 49 and 50.---Brown Horse-shoe Furnace.

The **Brown Horse-shoe Furnace**,\* shown in Figs. 48 to 50 may be considered as a Brown-Allen-O'Hara furnace with the upper hearth removed, and the lower hearth bent round into four-fifths of a circle. Only two stirrer carriages are used, and that which has finished making the circuit of the hearth, and makes its exit by lifting up the flap-door which closes the end of it, strikes the other carriage which has been cooling in the open space,

\* *E. and M. J.*, May 12, 1894; also *Peters. l.c.*, p. 218.

pushing it along until it becomes gripped by the cable, the first carriage being at the same time automatically disengaged, and standing to cool off until the circuit is made by the other carriage. These furnaces have been somewhat largely used on mattes, but not so much on ore; particulars of their work on both classes of material will be found in Table XII.

Each Horse-shoe furnace requires 35,000 lbs. ironwork, besides 20,000 lbs. of buckstaves, 60,000 ordinary bricks, 4,500 plain firebricks, and 25,000 lbs. weight of wedge brick and fireclay tiling, the total cost, f.o.r. Chicago, being about £1,500.

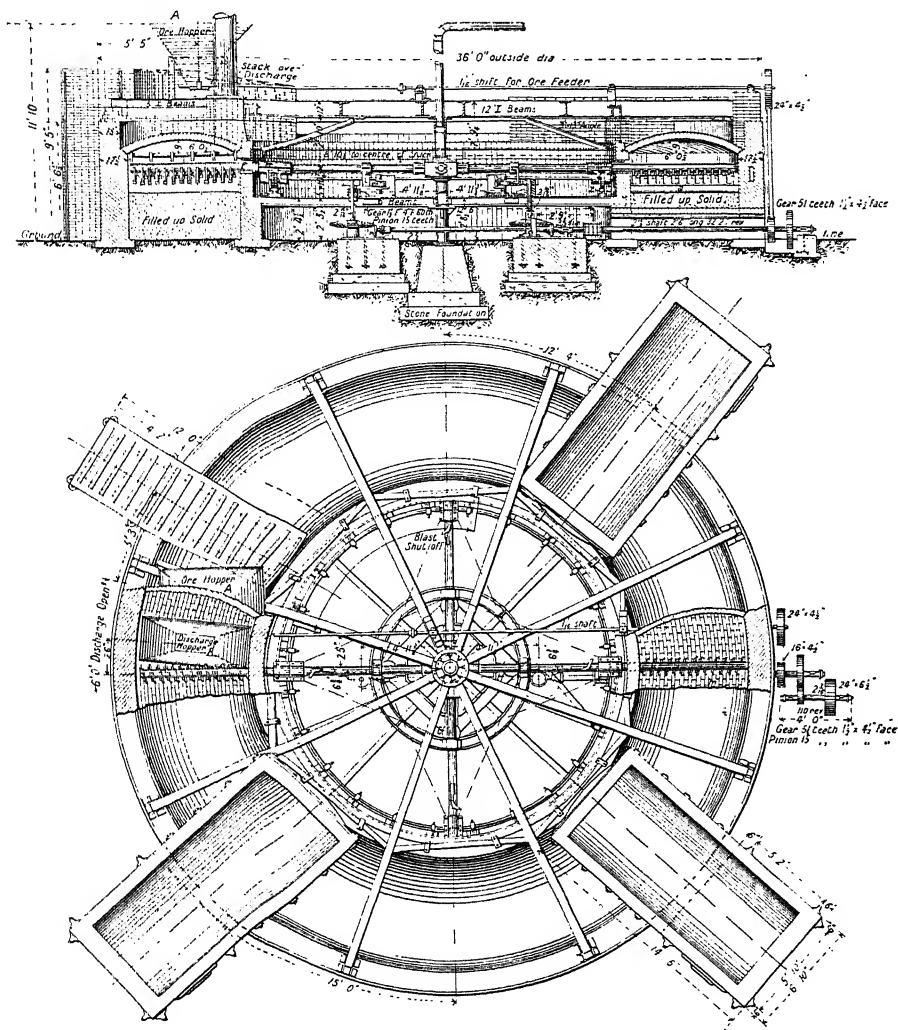
One of these furnaces was formerly at work at *Argentine* (Kas.) roasting lead-matte for the Hunt-Douglas process.\* The matte contained 10 to 20 per cent. Pb, 25 to 35 per cent. Cu, and was roasted at the rate of 18 tons per day, with a consumption of 20 per cent. by weight of slack, but the wet process referred to has now been abandoned.

**The Pearce "Turret" Furnace**, originally devised for roasting pyritic ores and concentrates, has been used for roasting leady ores, and more especially mattes, with fair results. The general construction of this furnace is shown in Figs. 51 and 52, from which it is seen to be a reverberatory hearth built in a circular form, with an open sector through which the discharge of the roasted ore takes place, and an open space inside the circle, making the actual working hearth annular in form. In the centre of the enclosed space a vertical column supports four horizontal radiating arms of 5-inch pipe, revolving with the column at any desired speed, to which are attached rabble blades for stirring the ore; only two of the arms, however, in practice usually carry rabbles. The pipes are perforated in front of each of the ploughs or rabbles, and air forced through the pipes escapes through the perforations, cooling the arms and rabbles, while at the same time supplying a current of hot air to promote oxidation and accelerate the roasting. To allow of a continuous slot in the inner wall of the circular hearth through which the arms must pass, all that side of the arch is supported by stirrups from eight 12-inch I beams arranged radially over the top of the furnace, the slot being closed between the arms by means of a travelling steel band. The crushed ore, fed at A by means of a hopper with adjustable mechanical feed, is pushed along the circle at any desired speed by the rabbles, and discharged automatically at B into iron tramwagons underneath. The hearth is heated by two stepgrate fireplaces automatically fed with slack coal, although in several cases oil is now used as fuel instead of coal. The space underneath the hearth was formerly utilised for dust chambers, but it is now found better to build the furnace solid and have the dust chambers independent. The rabble-blades have to be replaced every three to six weeks, beyond which the cost for repairs is very trifling. The cost of a 36-foot furnace in Denver is not much over £1,000, so that it is considerably cheaper than any other mechanical furnace in first cost. Each 6-foot furnace requires 40,000 lbs. of ironwork, besides 65,000 ordinary and 3,500 firebricks. One man only per shift is required to clean the grates and attend to the furnace.

On pyritic ores containing some silica and little or no lead the capacity of this furnace is from 15 to 20 tons per twenty-four hours, with a fuel consumption of 16 per cent. or under, and the total inclusive cost of roasting such ores in Colorado is only 3s. per short ton.

\* For which the volume on the *Metallurgy of Silver* should be consulted.

At the *Colorado* and *Philadelphia* Smelting Works (Pueblo, Colo.)\* furnaces of this pattern were erected and tried for roasting leady ores, for which work, however, they have not given much satisfaction on account of comparatively high fuel consumption, and of the large amount of flue dust made,



Figs. 51 and 52.—Pearce Turret Furnace.

although the actual cost of roasting is lower than in the hand-worked furnaces—namely, a trifle under 4s. per short ton. On ores running high in lead it has been found advisable to use only one of the rabble arms for stirring,

\* *Private Notes*, 1897 and 1902.

instead of two as used with ordinary ores and mattes, since with such ores it is important not to turn over ignited particles until the sulphur has been partly burnt off. The ore takes five to six hours to pass through the furnace, the daily output being 12 to 15 tons; one man per shift attends to two furnaces, but the coal consumption is nearly twice as much as with hand furnaces, and the amount of flue dust produced is from 2 to 3 to 4 per cent. by weight, as against one-half of 1 per cent. or less for the hand furnaces. Sintering on the hearth is combated by a heavy adjustable plough for tearing up the crusts, the invention of Rhodes.

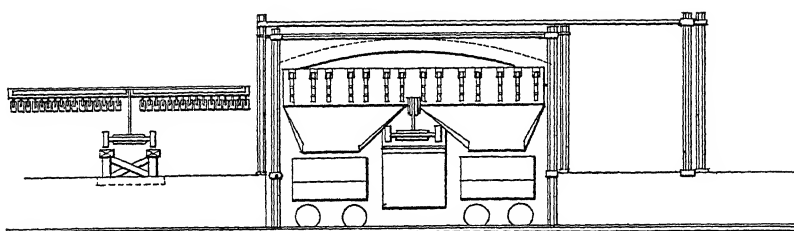
Although lead ores cannot be satisfactorily roasted in this furnace, mattes containing from 10 to 14 per cent. of lead as a maximum are successfully roasted, the sulphur contents being reduced from 20 or 21 per cent. down to 5 or 6 per cent. at the rate of 12 to 14 tons per day, and at a total inclusive cost which, including power, repairs, and depreciation, is stated to be under 3s. 6d. per short ton. Table XII. gives some comparative figures relating to the work done on mattes at Colorado plants. At *Selby* (Cal.)\* lead mattes containing 8 to 12 per cent. Pb are roasted in the Turret furnace down to 4 per cent. sulphur at the rate of 14 tons per twenty-four hours.

The **Ropp Straight-line Furnace**† has proved one of the most successful of all the mechanical furnaces for roasting leady material. Some of its details will be understood from reference to Figs. 53 to 56. Each of the six rabble-blades, which extend the full width of the hearth, is supported on a pair of four-wheeled trucks (shown in Figs. 55 and 56), and these, instead of running on a track level with the hearth, and only separated from it by means of a sheet-iron curtain (as in the Brown-Allen O'Hara furnace), run in a vault beneath the arch, connection being made by vertical arms which pass through a continuous cast-iron slot  $1\frac{1}{2}$  inches wide in the centre of the hearth. These arms are attached to a  $\frac{5}{8}$ -inch wire rope instead of to a chain, and while both rope and carriages are much better protected from the heat of the hearth than in the O'Hara furnace, they are more than half their time outside the hearth altogether, so that they get thoroughly cooled down at each revolution, which takes three and a half minutes. The ropes pass round two large sheaves at the furnace ends, and the track for the stirrer carriages bends round outside the furnace, as shown in the figures, so that the carriages run continuously with the rope without any jerking or irregularity of motion. The length of hearth in the ordinary or small-sized furnace is 105 feet by 11 feet wide in the clear, with four stirrer carriages and usually three fireplaces, arranged as shown in the figure. These fireplaces have air-cooled bridge walls, which serve the purpose of supplying heated air for oxidation. The ore is fed in from hoppers at one end of the furnace by means of Challenge automatic feeders, and is discharged into roasted ore hoppers at the other end. There is no complication of superposed hearths, and the stability of the principal arch is in no way interfered with by openings; the furnace, therefore, suffers less from racking and wear and tear than most others, while it is easily accessible for repairs. The ropes last about eighteen months, and the rabble-arms even longer; the rabble blades themselves last six months on the fire side, and double as long on the side opposite to the fireboxes.‡ There are three sizes of furnace, the smallest

\* Douglas, *Journ. Soc. Arts*, Aug. 16, 1895, p. 832.

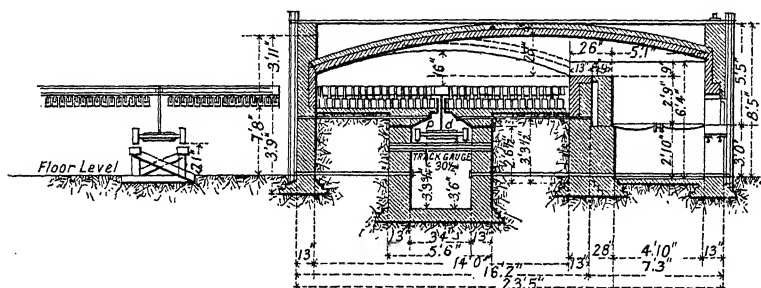






Discharge End Elevation

Fig. 55.



Section Through Fire Box A.B.

Note. All Fire Surfaces to be Lined with Fire Brick.

Fig. 56.

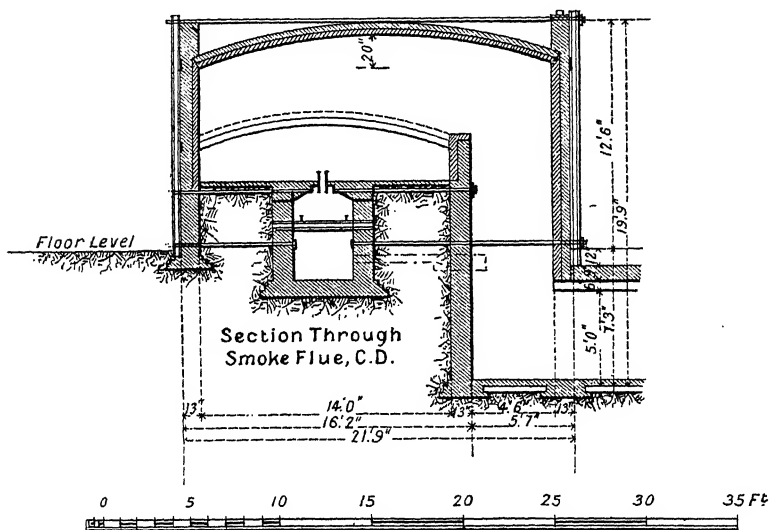


Fig. 56A.

Figs. 55 to 56A.—Ropp Straight-line Furnace.

measuring 105 feet  $\times$  11 feet, takes 5 to 6 H.P. to drive it, and burns  $2\frac{1}{2}$  to 3 tons of coal per day for 26 to 40 tons of material roasted; the largest size,

which is that figured, measuring 150 feet  $\times$  14 feet, takes 7 to 8 H.P., and roasts from 45 to 80 tons of material per day with a coal consumption of about 13 per cent. by weight. The largest-sized furnace contains 76,000 lbs. of ironwork, besides another 18,000 lbs. of buckstaves, or 94,000 lbs. in all: 164,000 ordinary and 29,000 firebricks are also required. The total cost of the ironwork alone is about £1,300, f.o.b. San Francisco.

Some data of the performance of the furnace at the *Selby Smelting Works* (Cal.), compared with the work done by the *Pearce Turret* furnace—both furnaces running side by side at Selby on the same ore—are given in Table XIII.

TABLE XIII.\*—COMPARISON BETWEEN PEARCE TURRET AND  
ROPP STRAIGHT-LINE FURNACE.

	Pearce Turret.	Ropp Straight-line.
Percentage of lead in ore, . . . . .	12 to 15 per cent.	
" of sulphur in raw ore, . . . . .	20 to 25 "	
" sulphur in roasted ore, . . . . .	5 per cent.	4 to 5 per cent.
Dimensions of furnace, . . . . .	36 feet diameter.	105 feet by 11 feet.
Approximate cost of furnace, . . . . .	£1,200.	£1,800.
Time spent in furnace by each particle of ore, . . . . .	Five hours.	Six hours.
Output of roasted ore per twenty-four hours, . . . . .	12 to 16 short tons.	25 to 35 short tons.
Mean consumption of coal, . . . . .	2 short tons.	4 short tons.
Percentage " " " " " " " " " " " "	15 per cent.	13 per cent.
Labour per furnace-day, . . . . .	Two men.	Two men.
H.P. required to drive, . . . . .	4 H.P.	5 to 6 H.P.
Repairs and depreciation, . . . . }	Considerable.	Very little.
Fine dust produced, . . . . }		

On these figures there would seem to be some saving in cost in favour of the "Straight-line," but evidently its principal advantages are simpler construction and better cooling of the rabble blades (which means less cost for repairs) and the smaller amount of flue dust produced. It is no doubt the use of a blast for cooling the rabbles which increases the amount of flue dust produced by the Turret furnace. As regards economy of fuel, neither furnace seems to reach a high standard, and no doubt any of the more complicated furnaces with superposed hearths would give better results in this respect if they could be depended upon to handle leady ores without clotting.

At the *Port Pirie* Works of the Broken Hill Proprietary Company five Ropp furnaces, 150 feet long  $\times$  14 feet wide, are in use for roasting lead concentrates. Formerly, when 55 per cent. concentrates were roasted alone, considerable difficulty was experienced through clotting and sintering; but now that to the concentrates additions are made of 14 per cent. by weight of shell sand,  $4\frac{1}{2}$  per cent. of fine iron ore, and 9 per cent. of siliceous ore, the tonnage put through daily has been increased to 100 tons per furnace without difficulty. Each furnace discharges into a hopper, from which are

\* *Private communication*, A. v. d. Ropp, Nov. 1896.

† Delprat, *E. and M. J.*, March 16, 1907, p. 516.

filled iron trucks holding a ton each, and these, by means of a hydraulic lift, are raised to the floor above the H.H. pots.

Particulars of work done by the Ropp furnace at the now dismantled *Hanauer Works* (Salt Lake City) are given in Table XII.

**Mechanical Reverberatories with Multiple Hearths.**—The best-known furnaces of this class are the *Spence*\* and *Keller*† furnaces. Both have superposed hearths, four in the *Spence*, five in the *Keller*, traversed by rakes having a reciprocating motion; but in the latter furnace, which is much more modern, the mechanism and stirrer carriages are altogether outside the furnace, only the rabble-arm itself entering the furnace by means of a slotted wall, the slot being made tight by a travelling steel band. A pair of furnaces are built 8 feet apart, so as to use only one set of carriages and mechanism for the two sets of rakes.

These furnaces do splendid work on pyritic ores, not only at Butte, where they were first introduced, but also at the *Germania Works*,‡ where, however, extensive experiments have proved that for good work in roasting the limit of lead in the ores is reached at about 10 per cent.; and further, that the furnace is not suitable for roasting lead matte owing to the difficulty of controlling its tendency to sinter on the upper hearths. Table XII. gives details of the work done by this furnace on ore containing 8 per cent. lead at the *Germania Works*.

The *Wethey* § furnace is another furnace with superposed hearths built in two blocks, having the mechanism and stirrer carriages running in the space between the blocks. It consists of four superposed hearths, each 50 feet long by 10 feet wide, heated by a firebox at the side, and the ore is worked forward from the top hearth to the bottom by the usual stirrers of rabble blades fixed to pipes resting on carriages, which run on tracks fixed outside the furnace. The peculiarity which distinguishes this from the *Keller* furnace is that the rabbles, instead of having a reciprocating motion and returning upon their tracks idle after tripping 90°, turn completely over and return upon the hearth immediately below, so completing the circuit and carrying the ore on each hearth continuously in one direction. The furnace is, therefore, essentially a four-hearth O'Hara, but much improved as to details.

Each furnace requires 170,000 lbs. of ironwork, 122,500 ordinary and 2,000 firebricks, and the cost at *Butte* is over £2,000.

Furnaces of this type at the *Butte Reduction Works*,|| working on zincy and coppery pyritic ores, treat 30 to 35 tons per twenty-four hours, bringing down the percentage of sulphur from 40 to 6 or 8 per cent.; they consume only 2 cwt. of slack coal to the ton, or 10 per cent. The cost for fuel and labour is only 1s. 6d. per ton, and the repairs in two years have been nominal in amount.

Of course, the results on lead ores would not be nearly so good as the above, yet it should be possible to work them; in fact, the author is informed that a furnace of this type is actually at work in an English works roasting galena concentrates from Broken Hill, but no details as to performance are available.

\* v. Peters, *Modern Copper Smelting*, 7th Ed., 1895, p. 220.

† Douglas, *Journ. Soc. Arts*, Aug. 16, 1895; Peters, *op. cit.*, pp. 214-218; also Hofman, *Metallurgy of Lead*, 1906, pp. 191-195.

‡ *Private Notes*, 1907.

§ v. Hofman, *op. cit.*, pp. 195-198.

|| *Private Notes*, 1897.

**Revolving Hearth Furnaces.**—These are all modifications of the old *Brunton* furnace, patented in 1828, and since largely employed for roasting arsenical tin ores. The rotating bed is a very flat cone, and the ore is stirred one or more times each revolution by a fixed plough or ploughs. The *Blake* furnace on this principle rests on cast-iron balls in an annular groove instead of upon a central vertical shaft. The *Heberlein* roasting furnace,\* so largely used in connection with the Huntington-Heberlein process to be described in the next chapter, is an improvement on the old Brunton calciner still largely used for roasting tin ores, being essentially a circular revolving hearth inside a fixed dome, with fixed fireplaces, air-ports, and rabble-arms. The hearth is 26 feet in diameter, and a down-turned flange at its periphery, dipping into sand in an annular trough, shuts off the combustion chamber from communication with the outside air, except through the ports designed for its admittance. The 26-foot hearth revolves about four times per minute, and takes about  $1\frac{1}{2}$  H.P. ; it roasts about 36 tons per twenty-four hours from 20 to 22 per cent. sulphur down to 10 to 11 per cent. with a consumption of

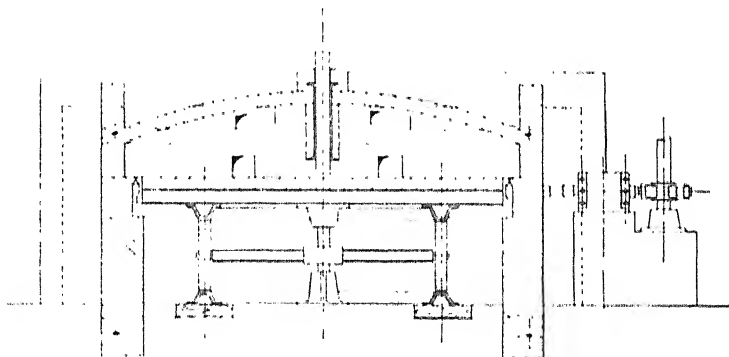


Fig. 57. Revolving Hearth Furnace (Heberlein Type) Section.

22.5 per cent. by weight of coal. The furnace is built up high above the ground, in order that the driving mechanism under the hearth may be conveniently got at, and in order to permit the ore to be discharged into suitable hoppers. One man on each shift of eight hours can attend to two furnaces. The cost of roasting is given in Table XII.

Figs. 57 and 58 show a large furnace on this principle which has air-cooled rabble-arms. The rabbles themselves are simple flat blades, the height of which in the arm is adjustable to allow for wear, and they are cooled by the powerful indraught of air around them through the roof, which also greatly facilitates oxidation, inasmuch as the current of fresh air is brought directly in contact with fresh surfaces of heated ore. The hearth figured is flat, and is driven by means of an ordinary drive-chain with sprocket wheels, but in a later and better modification the hearth is driven by means of large gear-wheels underneath, like the original Brunton.

At the *Sullivan Smelting Works* (E. Kootenay) two Heberlein furnaces

\* Ingalls, *Bull. Amer. Inst. Min. Engrs.*, Sept. 1906, p. 688.

treat 45 tons of charge each per twenty-four hours with a fuel consumption of 20 per cent. ; other particulars are given in Table XII.

At *Peñarroya* (Spain) six furnaces of this pattern serve to prepare material for blast roasting, the capacity of each furnace being 54 tons per twenty-four

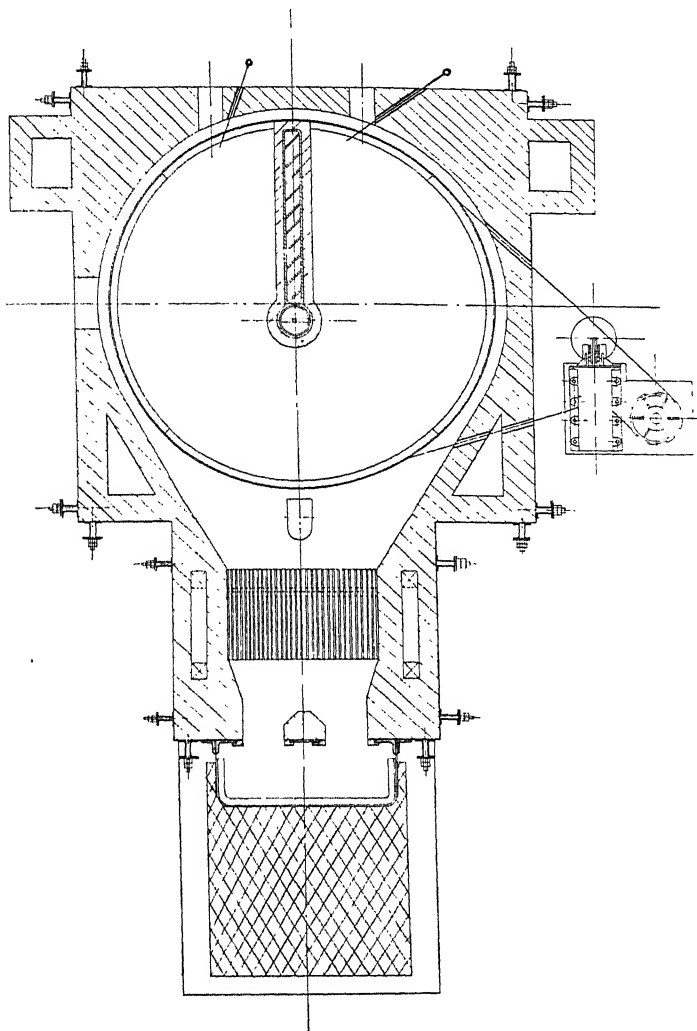


Fig. 58.—Revolving Hearth Furnace (Heberlein Type)—Plan.

hours of a mixture composed of 40 tons galena ore and 14 tons of “fluxes,” these consisting of slaked lime and crushed iron ore in varying proportions.

The **Godfrey**\* furnace, like the Brunton and the Heberlein, one in which the hearth revolves, while the rabbles are stationary ; but it differs from

\* For an illustration of this furnace see *Trans. Inst. Min. Met.*, vol. vii., p. 324.

them in two respects—namely, in having the hearth and the arch over it built annular instead of circular, resembling in this respect rather the Pearce Turret and Brown Horse-shoe furnaces; and in having, like the latter, a sector of the arch cut out, 3 feet 6 inches wide at the outer circumference and 2 feet wide near the centre, within which sector the single fixed plough works, so that it is always kept cool, and the hot ore gets exposed to fresh air at the moment when the rakes turn it over.

The hearth is supported by ten cast-iron cantilevers fixed into a central shaft or pillar, it is built of cast-iron plates upon which the working hearth is of 2-inch fire-tiles; its outside diameter is 20 feet and the inside diameter 4 feet 6 inches, the hearth thus being 7 feet 9 inches wide and 38 feet 6 inches in length on the median line. It is carried on ball-bearings, the balls or rollers running in a groove, owing to which the friction is so low that the power required for driving is only 1 H.P., and the usual speed is from 17 to 30 revolutions per hour. The ironwork for this furnace weighs 25 tons, and the furnace also requires 8,000 firebricks and 7,000 red bricks.

At a leading works in Australia these furnaces are in use for preliminary roasting of lead ores as a preparation for pot-roasting, but the driving gear has been somewhat modified so as to assimilate them to the Heberlein furnaces. Details of the results of these furnaces at the works referred to are given in Table XII., and it may be mentioned that the total cost of each furnace erected was £1,650.

At *Murray* (Utah)\* five of these furnaces, 26 feet in diameter, are at work roasting leady ores preparatory to blast roasting at the rate of 8 to 12 tons per furnace per twenty-four hours, bringing down the sulphur contents of the ore from 18-25 to 8-12 per cent. with a coal consumption of 3,500 to 4,500 lbs. (19 to 22 per cent. by weight).

**Revolving Cylinder Furnaces.** There are two classes of cylinder furnace—namely, that with horizontal axis and intermittent charging, a single charge being revolved until finished, the type of which is the *Brückner* furnace; and that with inclined axis and continuous charging, of which type are the *Oxland* and *White*† furnaces.

The *Oxland Calciner*.—Although giving excellent results with argentiferous mattes and pyritic copper ores, continuous cylinders have hitherto not come much into use for leady material, owing to the difficulty of controlling the temperature; and particularly to that of getting the heat high enough at the end where the ore enters, without overheating and sintering at the discharge end. These difficulties, however, are most felt where a dead roast is aimed at, and not so severely where only a partial roast is required as a preliminary to pot-roasting.

Figs. 59, 60, and 61 show an arrangement of an *Oxland* cylinder which has been in successful use at *Laurium* ‡ (Greece) for some years for partially roasting a mixture of lead ores and fluxes prior to pot-roasting. The cylinder is 42 feet 9 inches long by 3 feet 1 inch in diameter inside the firebrick lining, and revolves once in four minutes. It rests upon four sets of rollers, which are shown in the sections, but omitted from the elevation at the points *a*, *b*, *c*, and *d* for greater clearness. It is heated by a gas-producing fireplace

\* *Min. Ind.*, vol. xvi., p. 665.

† For details of which see the volume on the *Metallurgy of Silver* in this series.

‡ *Private communication*, 1909.

as shown, the fuel consumption being 5 per cent. of the weight of the charge. The hot calcines drop into the iron waggon shown, which delivers them direct into the H.H. pots. The lead concentrates treated at these works contain on an average Pb 60 to 62 per cent., Zn 8 per cent., Fe 7 per cent.,  $\text{SiO}_2$  2 per cent., S 18 to 22 per cent.; they are mixed with 20 per cent. by weight of crushed limestone and 15 to 20 per cent. by weight of siliceous lead carbonates. The resulting mixture loses 15 per cent. by weight in roasting, and after the completion of that operation contains 41 to 45 per cent. lead, 7 to 9 per cent.  $\text{CaO}$ , 7 to 8 per cent.  $\text{SiO}_2$ , and about 4 per cent. sulphur. The furnace puts through 30 metric tons per twenty-four hours. Other data referring to the work are given in Table XII.

The weight of ironwork for one furnace of this size is about  $18\frac{1}{2}$  metric tons, and the cost of ironwork only is about £588.

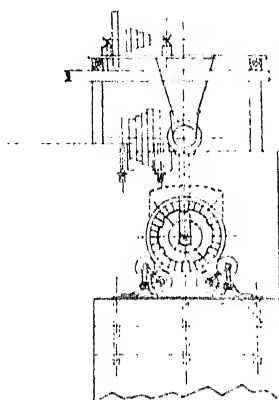


Fig. 59.

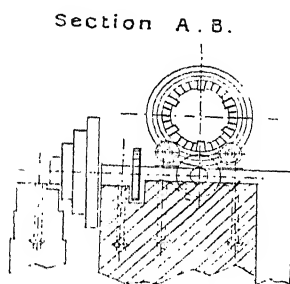


Fig. 60.

The **Brückner cylinder** in its modern form with separate firebox running on a track in front of the row of cylinders is shown in Figs. 62 and 63, has been adopted on a large scale for roasting pyritic ores in Montana, and has been applied to the roasting of leady ores at several works.

At the *Germania Works* (Utah)\* Brückner cylinders, 22 feet long by  $8\frac{1}{2}$  feet in diameter, gear-driven, were until recently used for roasting both ore and matte, and put through  $8\frac{1}{2}$  tons each per twenty-four hours of pyritiferous ores containing 20 per cent. lead, roasting down to  $2\frac{1}{2}$  or 3 per cent. sulphur, while ores with 30 per cent. lead roast down to 5 per cent. sulphur in the same time; the fuel consumption in each case being about 20 per cent. of coal by weight. Even ores containing as high as 35 per cent. lead can be successfully roasted by adopting special precautions to keep the temperature low, especially at the neck of the cylinder and at the beginning of the roast. The marked success of the large *Brückner* cylinders first introduced at these works, compared with results obtained elsewhere, is due partly to the unusually large diameter of the cylinders themselves, partly to greater care in regulating the firing, and partly to the adoption of special devices for equalising the temperature, consisting of steam-jets entering the

\* *Private Notes*, 1897.



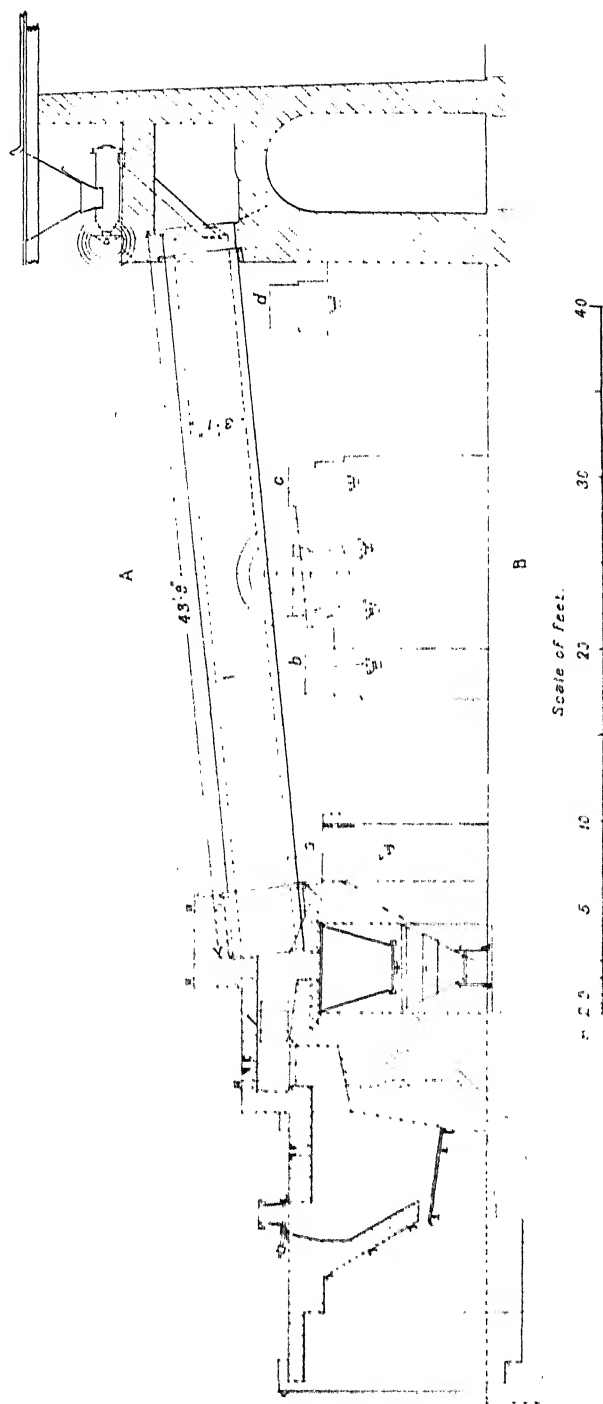


Fig. 61.—Oxland Furnace.

cylinder at the bottom of its neck and just below the neck of the firebox. By means of this device, and of rotating the cylinder at an unusually slow speed—namely, from 1 to 2 revolutions only per hour—sintering or “balling” of the charge becomes a comparatively rare occurrence, and even when this does happen, the form of the cylinder permits of ready access for “barring off.” After a new charge is dropped into the cylinder from the feed hopper, it is fired heavily for a short time, in order to raise the temperature to the point where roasting begins; the heat generated by the reactions is then generally sufficient to keep up the temperature for many hours, after which the cylinder has to be again fired to complete the roast. The total time required for roasting varies from 36 to 48, and in some cases even 60 hours. Some further particulars are given in Table XII.

One disadvantage of the Brückner cylinder for roasting ores rich in sulphur is the fact that the current of air does not come into such close contact with fresh surfaces of ore as is the case with mechanically stirred furnaces. It has been already remarked that in presence of a large excess of  $\text{SO}_2$  very little oxidation goes on, even when a considerable percentage of free oxygen is present in the gases; and the mass action of the large body of ore completely saturated throughout its pores with  $\text{SO}_2$ , which is only gradually removed by the central current of furnace gases, tends to retard very much the progress of the roast, especially at first. Besides cooling the neck, therefore, where the tendency to sintering is strongest, jets of steam or compressed air have an important effect in stirring up the stagnant sulphurous gases playing about over the surface of the ore, and driving them into the flue.

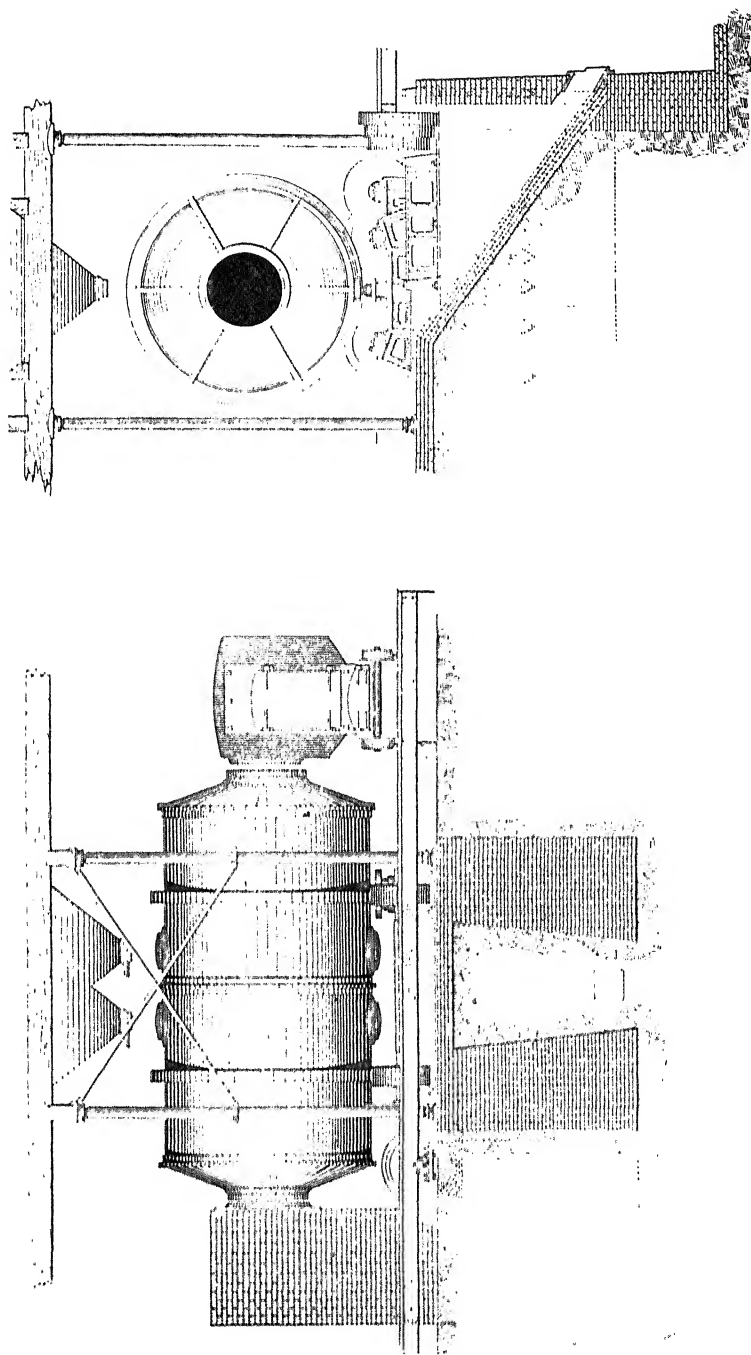
Another disadvantage of the Brückner, and indeed of all revolving cylinders, is the very high proportion of flue dust made, which is rarely less than 5 and may be as much as 10 or even 15 per cent. by weight of the ore. Repairs also are heavy, on account of the frequent necessity for barring off crusts, although they can be lessened by building the lining entirely of firebrick; while, except in the hands of a really skilful foreman, frequent trouble is sure to be experienced through “balling” and irregular roasting.

Following the example set by the Germania Works, Brückner cylinders have been erected at several of the smelting works in Colorado and Utah. The following figures\* show results obtained at three large works:

Works.	No. of Furnaces.	Minutes per Rev.	Weight of Charge.	No. of Hours per Charge.	Per cent. S in Roast.	Consumption of Fuel.
Pueblo, . . .	5	50	24 tons.	48 60	6	..
Globe, . . .	12	60	24 „	70 90	..	..
Murray, . . .	20	40	24 „	48	16	16.7

**Shaft Furnaces.** The only furnace of this kind which has been successfully used on pyritous lead ores is the well-known *Gerstenhofer*,† which at Freiberg puts through in twenty-four hours about 60 tons of a pyritous ore containing 18 per cent. galena, 60 per cent. pyrites, and 22 per cent. gangue, and roasts it down to about 7 per cent. sulphur. Ordinary lead ores, properly so called, could not, of course, be roasted in such a furnace at all.

\* From *Mineral Industry*, vol. xiv., pp. 396-7, and vol. xvi., p. 665.



Figs. 62 and 63.—Brückner Cylinder with Separate Firebox.

**Muffle Furnaces.**—The same remark applies to the use of muffle furnaces for roasting ores high in lead. At Oker, the *Hasenclever*\* muffle furnace is employed for the first roasting of mixed ore fines which average 11 per cent. galena, 15 per cent. chalcopyrite, 28 per cent. blende, 25 per cent. pyrites, 14 per cent. heavy spar, and 7 per cent. other gangue matter; but this is only because of the special necessity which there exists of collecting and condensing the sulphurous gases for the manufacture of sulphuric acid.

**Summary.**—As a rule, it may be said that for rich galena ores containing but little of other sulphides, except blende, there is at present—apart from blast-roasting—no alternative to the use of hand-rabbed long reverberatories, for which all coarse particles should first be crushed to a 6- and preferably to an 8- or 12-mesh.

For a partial roast of such material rich in lead preparatory to agglomeration in pots, the *Heberlein* and *Godfrey* furnaces seem to have found a good deal of favour.

Where the ores are poorer in lead, or where the percentage of this metal can be reduced by the admixture of pyritous or other fluxing material to not over 20 to 30 per cent., especially when labour is costly, a mechanically rabbed furnace like the *Ropp* seems to be, in view of its simplicity, well adapted to the conditions. For mattes or ore mixtures containing not over 15 per cent. lead, where the quantity to be treated is not very large, the *Pearce* furnace may also be employed, in spite of its somewhat higher fuel cost and the higher percentage of flue dust made, which would appear to render the *Ropp* preferable for large installations.

When, however, iron pyrites is the predominating constituent of the ore, and the percentage of lead does not exceed 10 per cent., any of the mechanically-rabbed reverberatories may be employed for the ore fines, the multiple-hearth *Wetley* and *Keller* types, possessing decided advantages over the others, on account of their economy of fuel, though ill adapted to the handling of readily sintered material, even with special precautions to keep down the temperature.

As regards coarse ore, in which pyrites is the predominating constituent (say all of nut size or over), it is usually best crushed and roasted with the fines and with fine lead ore in a hand or mechanically rabbed reverberatory; although where local conditions permit it may be roasted in stalls or kilns at a lower cost, and occasionally to much greater advantage. The prices of labour, materials, fuel, and motive power, value of acid fumes, damage or nuisance caused by them, &c., must be duly considered, and each case decided on its merits.

It may be remarked with regard to all mechanical furnaces that they need careful handling by skilled labour, which is always expensive, and frequently not obtainable outside of great metallurgical centres; furthermore, as has been already seen, they are not adaptable to changes in the composition of the charge, and require a fairly uniform charge-mixture. Unless, therefore, the smelting works is on a large scale and favourably situated (as large-scale works generally are), both as regards the supply of reliable skilled labour and as regards a regular supply of ores of fairly uniform composition, it will usually be best to rely upon the ordinary hand-worked reverberatory for roasting all fine ores, except those containing so little lead

\* Schnabel, *op. cit.*, p. 83.

and so much pyrites as to render them ideal material for mechanically rabbled furnaces.

In view of the costliness of hand roasting, and, at the same time, of the absolute necessity for expelling the sulphur contents of a large part of the ores received, until within the past three or four years the use of mechanical furnaces was spreading greatly in large custom smelting establishments, and efforts were made to apply them successfully even for roasting material with a considerable percentage of lead. Since the general introduction of blast-roasting, however (for a description of which the next chapter should be consulted), and in particular since the development of that modification of the process in which ore mixtures only moderately rich in lead are roasted without the addition of lime, the pyritic and other non-lead material present serving to protect the charge from premature and too thorough fusion, ordinary roasting of fine ores plays but a secondary part in the economy of most large lead smelting establishments; and indeed its use is frequently restricted to the roasting of mattes, coppery pyritic ores, &c., all of which material is better suited to roasting in mechanical furnaces than are the lead ores proper.

## CHAPTER VI.

## BLAST-ROASTING OF LEAD ORES.

BLAST-ROASTING, more commonly although less appropriately called pot-roasting, is a comparatively new process, which combines the function of elimination of sulphur with that of agglomeration. It is a sinter-roast carried out usually in an iron converter or "pot," instead of a furnace, through the agency of an air-blast instead of an oxidising flame, and utilising as fuel the sulphur contents of the ore itself. The oxidation and generation of heat are so violent that, in view of the ready fusibility of compounds of lead, it is usual to submit the ore, if rich in lead, to a preliminary partial roast, in order to diminish the risk of premature agglomeration.

There are three main variants of the process :—

1. The original *Huntington-Heberlein process*, which comprises two distinct operations :—

A. Slowly heating a mixture of crushed galena with from 6 to 15 per cent. of burnt lime or crushed limestone in a reverberatory furnace to a temperature of 700° C., and then cooling to 550° C., and allowing free access of air, by which means about 25 to 30 per cent. of the lead present is converted to  $\text{PbSO}_4$  without any reduction to metallic lead. The operations of heating and cooling with free access of air are repeated until all but about 5 per cent. of the  $\text{PbS}$  is converted into sulphate with a little oxide. Up to this point the lime is stated by the inventors to be mostly unchanged,\* having served chiefly as a carrier of oxygen.

B. The hot partially roasted mixture is transferred to a converter, and blown up with an air blast under less than 8 ozs. pressure. The temperature rises immediately, owing to the oxidation of the residual sulphide, dense fumes of  $\text{SO}_2$  and  $\text{SO}_3$  are given off from the mass, which at first becomes pasty and then solidifies again. When evolution of  $\text{SO}_2$  ceases, the residue consists of a mixture of lead oxide and silicate and calcium sulphate.

2. The *Bradford-Carmichael process*, in which raw ore having a sulphur content of 13 to 17 per cent. is mixed with dehydrated gypsum, moistened, allowed to "set," broken up to 2-inch size, and fed into a small converter upon a little glowing fuel, where it is blown up by an air-blast. After the expulsion of steam, dense fumes of  $\text{SO}_2$  are given off, which are utilised for the manufacture of sulphuric acid.

3. The *Savelsberg process*, in which a mixture of raw ore and finely crushed limestone is moistened and charged gradually upon a bed of glowing fuel under a light blast until the converter is filled, when desulphurisation becomes active throughout the charge upward, and is closely followed by agglomeration. In this case also the gases, although not quite so rich in  $\text{SO}_2$  as those

\* This statement, however, will be seen to be incorrect.

from the Bradford-Carmichael process, can be utilised for the manufacture of sulphuric acid.

In all the above variants, the addition of lime in some form or other, whether slaked as sulphate or as limestone, is an essential feature, and, therefore, the generic name of "lime-roasting," first suggested by Ingalls, has been applied to them all. It appears, however, that the addition of lime in some form, although of great convenience, is by no means essential; and, therefore, all the lime-roasting processes may be regarded as only varieties of a more general method which is characterised by the use of a blast, and is generally carried on in pots or converters, and to which, therefore, the generic names of "blast-roasting" and "pot-roasting" seem to be suitably applied.

*Advantages of Lime-roasting and of Blast-roasting generally.*—As compared with ordinary reverberatory roasting, we may note :—

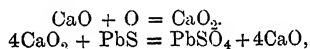
1. The plant required for a given roasting capacity is much less expensive and covers a much smaller space than for ordinary reverberatory roasting.
2. Volatilisation losses are lower, owing to the lower temperatures.
3. Repairs to furnaces and tools are less, for the same reason.
4. The product is in much better condition for blast-furnace treatment, being quite porous although agglomerated; the capacity of the blast-furnace department is, therefore, increased, and the cost of blast-furnace smelting reduced.

5. Any flux required by the ore (limestone or iron ore) can be mixed in a fine condition with the ore charged into the converters; such flux additions serve the double purpose of retarding and, therefore, regulating the reactions in the converter, and of partially fluxing the ore before it reaches the blast furnace, whereby again the capacity of the latter is increased, and the cost of working reduced.

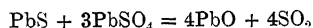
The increase in the capacity of the blast-furnace department consequent on the adoption of lime-roasting may amount to from 60 to 100 per cent.; thus at Port Pirie the same quantity of ore which formerly required thirteen furnaces, after the introduction of lime-roasting was smelted in seven, and similar results are reported from elsewhere. The increase in blast-furnace capacity results from three sources, namely :—

1. Coarser condition of the charge, the lower proportion of fine material always causing the furnaces to run faster.
2. More intimate admixture of ore and flux, which has the same effect.
3. More perfect expulsion of sulphur results in yielding less matte to be fed over again, and so leaves more room for ore.

*Reactions of Lime-roasting.*—As may be seen from an examination of their patent specifications, the originators of lime-roasting, Messrs. Huntington and Heberlein, thought that calcium peroxide played an essential part in the process, and gave the reactions

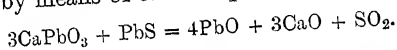


followed by



as covering the chemistry of the process. This explanation, however, ignores the fact that calcium peroxide could not possibly be produced at a red heat, or exist in presence of excess of  $\text{SO}_2$ .

Borchers\* found that calcium plumbate, produced by heating litharge with slaked lime, had a powerful influence in decomposing and oxidising galena, and believed that the principal reaction of the converter was the direct oxidation of galena by means of calcium plumbate.

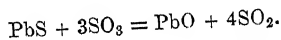


Calcium plumbate is, however, dissociated at temperatures of  $900^\circ$  to  $1,000^\circ$  C., so that it is hardly possible to assume its formation under the conditions prevailing in the roasting-pot, and certainly it could not be regenerated as this writer supposed; moreover, in the Carmichael-Bradford and Savelsberg processes, calcium plumbate could certainly not be formed, in the former case because of the excess of  $\text{SO}_2$  and  $\text{SO}_3$  present, and in the latter because the temperature at which the  $\text{CO}_2$  is expelled is about the same as that at which calcium plumbate formed by the action of litharge on quicklime begins to be decomposed.

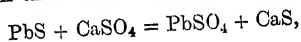
Hutchings† points out that a fully roasted charge always contains most of the added lime in the condition of  $\text{CaSO}_4$ , extractable as such with boiling water, and that this result is invariably attained whether the lime be added in the first instance as  $\text{CaO}$  or as  $\text{CaCO}_3$ . He looks upon the reactions in the converter as taking place in two stages.

1.  $\text{CaSO}_4$ , whether added in that form (Bradford-Carmichael) or formed *in situ* from  $\text{CaO}$ ,  $\text{SO}_2$ , and  $\text{O}$  (Huntington-Heberlein and Savelsberg), acts catalytically as a strong carrier of oxygen, the result being that the residual  $\text{PbS}$  is rapidly converted into  $\text{PbSO}_4$  with considerable evolution of heat.

2. As the heat rises the  $\text{PbSO}_4$  is acted upon by  $\text{SiO}_2$ , forming lead silicate with evolution of  $\text{SO}_3$ , which being red hot and, where generated, under slight pressure, acts energetically upon any  $\text{PbS}$  still remaining, according to the equation—



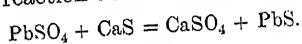
Doeltz‡ points out that the reaction supposed by Carmichael and others to play a principal part in the lime-roasting process, namely—



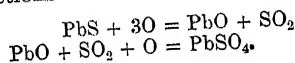
is endothermic, the thermochemical equation being

$$-17.8 - 318.4 + 216.2 + 92 = -28 \text{ cal.},$$

and proves by experiment that at  $400^\circ$  C. in a current of  $\text{N}$  or of  $\text{CO}_2$  the above reaction does not take place, but that, on the contrary, under the above conditions, the converse reaction does come into play, namely:—



He suggests, therefore, that  $\text{CaSO}_4$  plays no part in the process chemically, and that the only reactions which take place are the following:—



Guillemain§ remarks that other substances besides lime, such as crushed

\* *B. & M. J.*, Sept. 2, 1905.

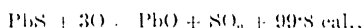
† *E. & M. J.*, Oct. 21, 1905, p. 726.

‡ *E. & M. J.*, Jan. 27, 1906, p. 175, translated from *Metallurgie*, vol. ii., No. 19.

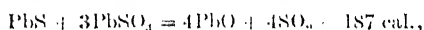
§ *E. & M. J.*, March 10, 1906, translated from *Metallurgie*, vol. ii., No. 18, Sept. 22, 1905 p. 433.



quartz, iron ore, or slags, none of which can be suspected of any chemical action in the process, can yet be made to serve the function of arresting the premature fusion of the lead sulphide and oxide before oxidation has been carried sufficiently far, and points out that if the strongly exothermic reaction—

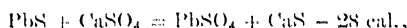


can be combined simultaneously with the endothermic reaction—

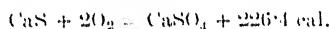


the process can be carried on successfully without any intervention of lime.

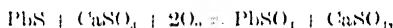
The author's view is that, while the endothermic reaction—



has been demonstrated impossible in a neutral atmosphere, the presence of oxygen, enabling the CaS to re-oxidise itself to CaSO<sub>4</sub> with evolution of heat, completely alters the conditions, for the combination of the reaction—



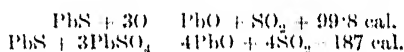
with that given above reverses the heat balance. We may then, perhaps, imagine the CaSO<sub>4</sub> to act by giving up its oxygen to the PbS, and becoming reduced to CaS, which in the act of formation is re-oxidised to CaSO<sub>4</sub> by the blast, according to the equation above. The nett result, expressed in one single equation—



is strongly exothermic, as shown by the thermochemical equation—

$$-17.8 - 318.4 + 216.2 + 318.4 = +198.4.$$

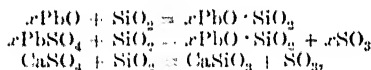
It seems quite probable, therefore, that the addition of lime to the converter charge does facilitate the oxidation of lead sulphide, yet, as pointed out by Guillemin, the addition is by no means essential, and the result of eliminating the sulphur and converting all of the lead into oxide can be attained by a combination of the exothermic and endothermic reactions—



premature fusion of the charge being sufficiently avoided by the addition of fine iron flux, quartz ore, or other inert substances.

As will be seen later, modern practice tends in the direction of doing away with the addition of lime, at some sacrifice of efficiency in the oxidation, and of regularity in the results obtained.

For the sake of completeness, it should be mentioned that, in addition to the above reactions, towards the end of the process the following also come into play :—



and, in the Carmichael-Bradford process in particular, it is chiefly the last of these reactions which so raises the percentage of sulphur in the gases evolved as to render the manufacture of sulphuric acid economical.

The fact demonstrated by Hofman\* that calcium ferrite begins to be formed by the action of  $\text{Fe}_2\text{O}_3$  upon  $\text{CaO}$  at a temperature of only  $1,000^\circ$ , and that it fuses at  $1,250^\circ$ , has some bearing upon the pot-roasting of lead ores to which both lime and iron oxide ores are added before roasting, as is now a common practice. The formation of this fusible ferrite undoubtedly facilitates subsequent slag formation in the blast furnace, and its high fusion temperature, compared with that of the silicates of lead, prevents premature agglomeration before the sulphides have become sufficiently oxidised. For rich galenas, therefore, which are to be pot-roasted and agglomerated prior to being added to the general smelting charge of a blast furnace, it is far more advantageous to mix them with fine iron ores rather than with so much silica,

**The Huntington-Heberlein Process.**—This process from Pertusola has extended its sphere to Mexico, Spain, Australia, Germany, the United States, and Canada; in fact, to almost every lead-smelting country. It comprises, as has been already seen, two stages, the first of which is a simple partial roasting, following an admixture of lime and sometimes of iron oxide or other slag-forming material, in mechanical roasting furnaces of various types. Some details of the roasting operation have been given in Chap. v. The partially roasted ore is cooled and transferred by conveyors or hopper trucks to hopper bins situated above the converters, a small proportion of it, however, being kept red hot in a separate bin, in order to start the reactions of each charge. The remainder of the ore mixture is damped, in order to prevent as far as possible overheating, and so equalise the progress of the operation.

The converters were originally conical in form, and of only 1 ton capacity, but they have been gradually increased in size, and now generally take the form of shallow bowls, often 9 feet in diameter and 4 feet deep, and of as much as 15 tons capacity. For the largest of these pots hand-turning gear is not considered adequate, and a travelling crane is substituted, which lifts them bodily and carries them to a dumping pit.

As a rule, the sulphur contents of the ore are reduced only to  $2\frac{1}{2}$  to  $3\frac{1}{2}$  per cent., because it is found that such an amount of residual sulphur gives no more trouble in the subsequent smelting operation than if it had been reduced by long-continued blowing to 1 per cent. This may, perhaps, be due to the fact that the residual sulphur exists chiefly in the form of calcium sulphate, from which it is expelled as  $\text{SO}_3$  in the blast furnace through the action of silica.

A new charge is started by kindling a little fire of coal or wood upon the grate, and adding a cwt. or so of hot roasted ore; a light blast (about 2 ozs.) is then turned on, and the charge of damp ore gradually added and spread in layers. When full, the blast pressure is increased to about 16 ozs., and if the operation is proceeding properly smoke rises evenly from the whole surface of the charge, the top of which remains quite cool until near the end of the operation, hence the volatilisation losses are quite low. If blowholes form, they must be stopped by ramming in damp ore. At the end of the operation the top of the charge shows dark red all over, with brighter spots at the blowholes, through which the blast escapes.

When the pot is tipped the charge, if properly roasted, comes out as a solid cake, which in falling to the ground or on to cast-iron cones placed to receive it, breaks up into a few large pieces, which have to be further broken

\* *Bull. Am. I.M.E.*, Jan. 1909, p. 69.

up by wedging and sledging. The time taken to blow a charge varies, partly owing to its size, partly owing to its composition, and partly to the form of the pot, between five and eighteen hours. Owing to the oxidation of lead and iron in the charge, and to the fact that the residual sulphur is in the condition of sulphate, there is a considerable increase of weight in the preliminary partial roasting in the furnace; but with the expulsion of practically all the sulphur as  $\text{SO}_3$  comes a reduction in weight, and the finished charge weighs less than did the original.

The conical cast-iron pots of the original H.H. process held little over a ton apiece, but these were soon enlarged to a size holding  $1\frac{1}{2}$  to 2 tons, which,

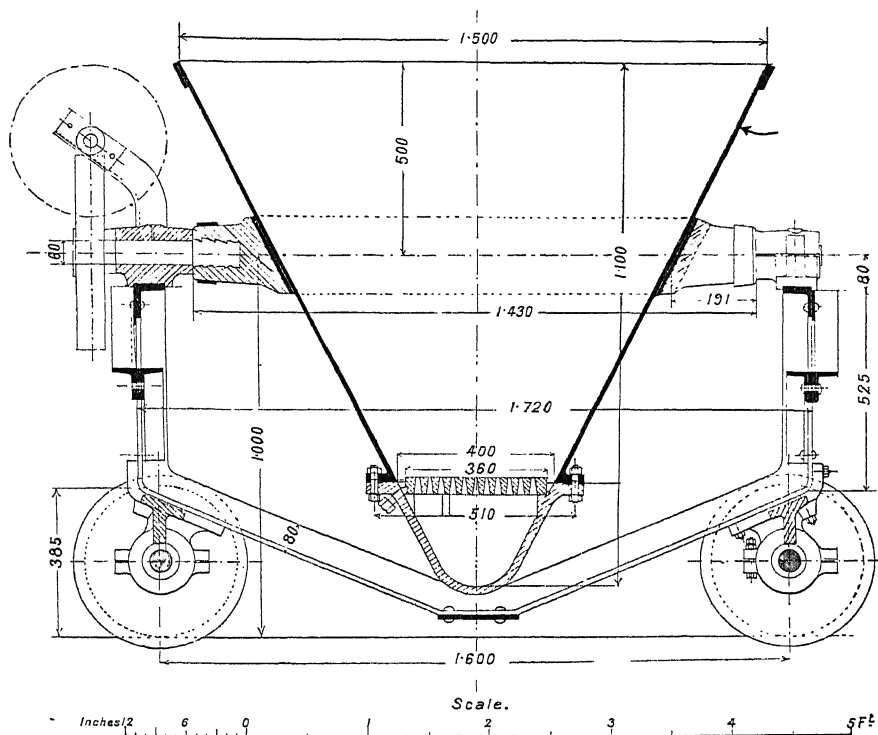


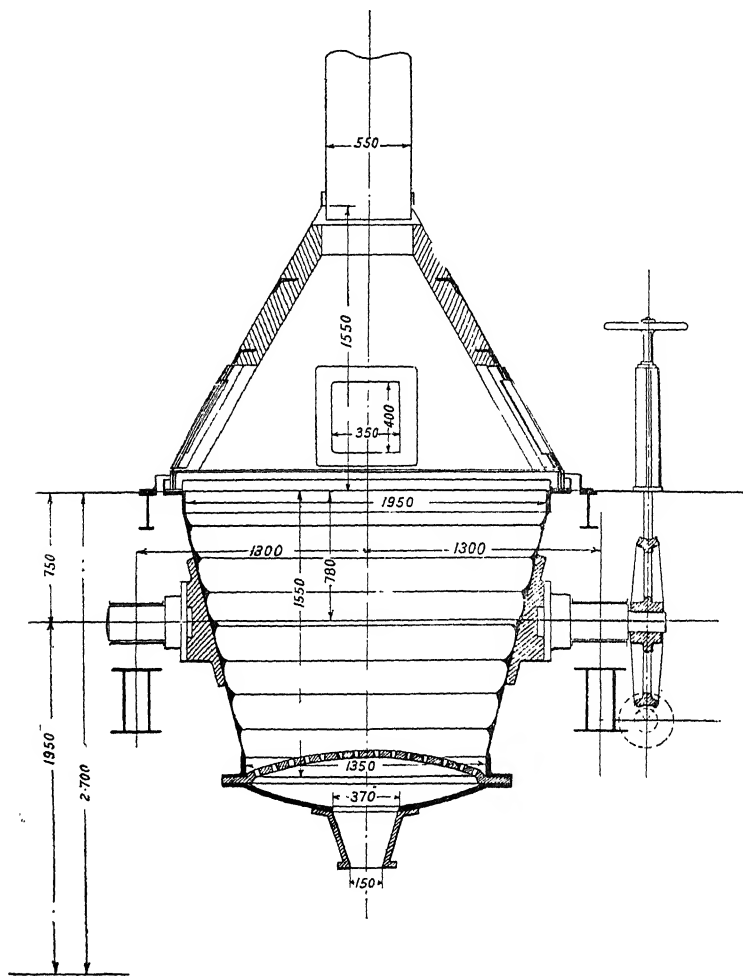
Fig. 64.

made of wrought-iron plate, is a very handy pot for hand-working with cheap labour. Fig. 64 shows such a pot in use at Laurium, the capacity of which is about 2 tons.

These small pots, however, are rapidly becoming obsolete for works handling large tonnages, especially where labour is dear, being replaced by much larger pots holding 8 to 10 tons. Fig. 64*a* shows a form of pot in use at some Continental plants, which holds 8 tons of average ore mixture.

At *Peñarroya*, six large revolving calciners of the Heberlein pattern serve to prepare the charges for the roasting-pots or converters. Each calciner

handles 54 tons per twenty-four hours, consisting of 40 tons of galena and 14 tons of "fluxes," composed of slaked lime and crushed iron ore in varying proportions, the mixture having been previously made up in convenient beds. The hot calciners from the preliminary roast are received in iron waggons and dumped into the conical pots, of which there are 70, holding over a ton each.



Dimensions in millimetres.

Fig. 64A.—8-ton Roasting-pot.

Above the hot charge is added cold half-burnt portions from previous charges well damped down, and blowing with a light blast lasts from  $3\frac{1}{2}$  to 4 hours. About one-half to two-thirds only of the roasted material is in the form of agglomerated lumps, the remainder being fines.

At the works of the *Tasmania Smelting Coy., Zeehan*,\* twelve 2-ton conical converters of plate iron, 5 feet 6 inches in diameter by 5 feet deep, hung on trunnions, and turned by hand gear, serve to treat the partially roasted mixture of galena, limestone, and slag-forming material from two mechanical roasters of 15 feet diameter, the air pressure employed at the finish being 17 ozs. per square inch.

At *Cockle Creek* similar converters are in use for treating the roasted product of the Godfrey roasters, which are fed with a mixture of fine galena concentrates, lime, and siliceous ore, so made up as to yield a product with Pb 20 to 25 per cent., CaO 6 to 9 per cent., SiO<sub>2</sub> 20 per cent., and S 6½ to 8 per cent. The blow lasts five hours, and the air pressure at the finish reaches 20 ozs.

At *Port Pirie* a battery of 17 converters handles the roasted ore mixture from the five Ropp furnaces mentioned in Chap. v. The converters are of cast iron, cast in segments, and hung on trunnions, they measure 8 feet in diameter and 6 feet deep, and the charge for each is 8 tons. At 2 feet from the bottom a perforated plate with a short cone upon it serves as a wind box, and the hood has a telescopic pipe to allow of raising, by means of a block and tackle, to allow the converter to be tipped. The blowing lasts four hours with air at 24 ozs. pressure, and the sulphur contents of the agglomerated ore are reduced to about 3 per cent.

In American works, according to Ingalls,† the standard-sized converter is a nearly hemispherical pot, 9 feet in diameter and 4 feet deep, 2½ inches thick at the bottom, thinning off to 1½ inches at the rim, supported on trunnions, and having a 6-inch blast pipe at the bottom, above which is a circular iron grate, 6 feet in diameter, ¾ inch thick, and provided with ¾-inch holes at 2-inch centres. Such a pot holds about 8 short tons of an ore mixture weighing 166 lbs. per cubic foot, and containing, say, 17 per cent. sulphur, which is roasted down to 3 to 5 per cent. in a period of 12 to 18 hours. Each converter requires about 100 cubic feet of air per minute, the pressure varying according to the stage of the process, but supposing a maximum of 16 ozs., the total power required for a battery of six pots treating 72 short tons of charge mixture, or, say, 58 short tons of ore per day, would only be 21 H.P. The cost of the converting process on the above basis is approximately as follows:—

	S.	D.
Labour 3 Foremen at 13/4 per shift, . . . .	40	0
„ 9 Men at 10/6 „ . . . .	94	6
Power—21 H.P. at, say, 1/3, . . . .	26	3
Supplies—Repairs and renewals, . . . .	21	0
	<hr/>	
Total cost on 72 short tons charge, . . . .	181	9

This is equal to 2s. 6d. per short ton, or 2s. 9½d. per long ton; on the original ore it amounts to practically 3s. 6d. per ton. The total cost of Huntington-Heberlein treatment up to the point of delivering the agglomerated material to the smelting department is approximately as follows:—

\* D. Clark, *E. and M. J.*, Oct. 20, 1904, p. 630.

† *Trans. Am. I.M.E.*, July, 1906.

	Per Ton Charge.	
	S.	D.
Mixing ore, lime, &c., . . . .	0	5
Preliminary roasting, . . . .	3	0
Delivering to converters, . . . .	0	6
Converting as above, . . . .	2	10
Breaking-up product, . . . .	2	3

9 0 = 11s. 3d. per ton ore.

This is quite as much as the cost of plain roasting in hand-worked reverberatories, or perhaps a little more, but, as has been already seen, the product is in a very much better condition, and the losses from volatilisation and dusting are much lower.

At *East Helena* (Mont.) Cover d'Alene concentrates mixed with other ores down to an average lead content of 40 per cent. are roasted in twelve Godfrey roasters, and the resulting half-roasted material with 25 per cent. lime is pot-roasted in large H.H. conical pots, this combination of Godfrey roasters and H.H. pots being preferred to all others. Nearly all the lead concentrates are thus roasted, because the use of raw concentrates in the blast-furnace charge is found to increase the matte-fall, and make foul slags.

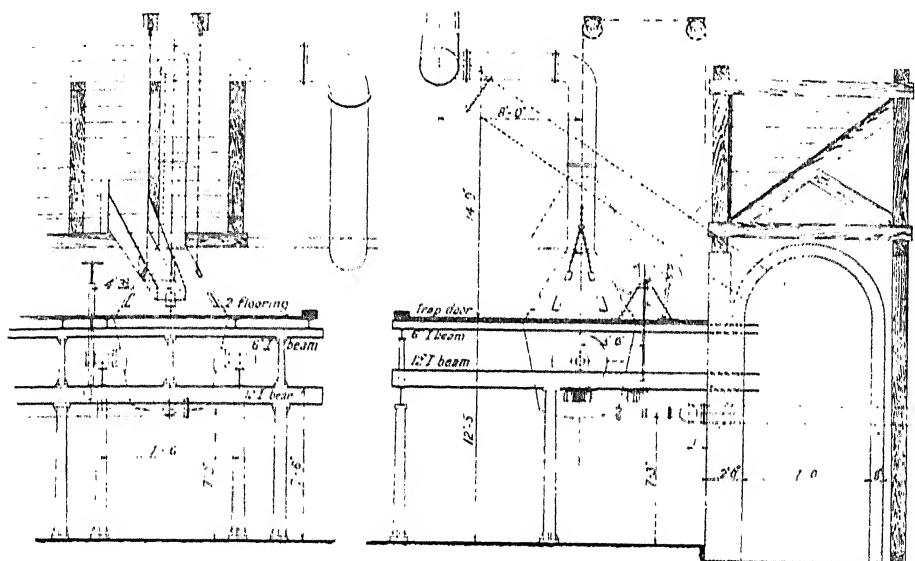
**The Carmichael-Bradford Process.\***—The essential points of difference between this process and the Huntington-Heberlein are two—namely, the omission of a preliminary partial roast and the use of gypsum instead of lime. Part of the sulphur contents of the gypsum as well as of the ore is driven off by the blast, and as a consequence the fumes are richer in  $\text{SO}_3$  than those of the H.H. process, and they are advantageously utilised for the manufacture of sulphuric acid. It is not often that a cheap supply of gypsum can be found in the neighbourhood of lead ores, and, therefore, the scope of the Carmichael-Bradford process is somewhat limited, but it has proved very useful at Broken Hill, although at the time of writing it does not appear to have entirely superseded the H.H. process at Port Pirie. The material treated at Broken Hill is a mixture of three parts lead slime from the settling tanks of the concentration mill and one part of concentrate, with one part of impure gypsum, and the general average composition of the charge mixture is as follows:—

Galena, $\text{PbS}$ , . . . . .	29
Blende, $\text{ZnS}$ , . . . . .	21
Pyrites, $\text{FeS}_2$ , . . . . .	2
Ferric oxide, $\text{Fe}_2\text{O}_3$ , . . . . .	2.5
Ferrous oxide, $\text{FeO}$ , . . . . .	1
Manganese oxide, $\text{MnO}$ , . . . . .	5
Alumina, $\text{Al}_2\text{O}_3$ , . . . . .	3
Lime, $\text{CaO}$ , . . . . .	10
Silica, $\text{SiO}_2$ , . . . . .	14
Sulphuric anhydride, $\text{SO}_3$ , . . . . .	12
	<hr/>
	99.5

The crude gypsum is first pulverised by a disintegrator, and then carried by means of a pan conveyor at a speed of 1 foot per minute through a dehydrating furnace, 20 feet long, after which it is elevated to a storage-bin on the same level as the ore storage-bin. Both ore bins and gypsum bins are provided with screw conveyors, which deliver automatically the desired proportions of

\* D. Clark, *E. and M.J.*, Nov. 3, 1904; W. R. Ingalls, *ibid.*, Oct. 28, 1905, p. 778.

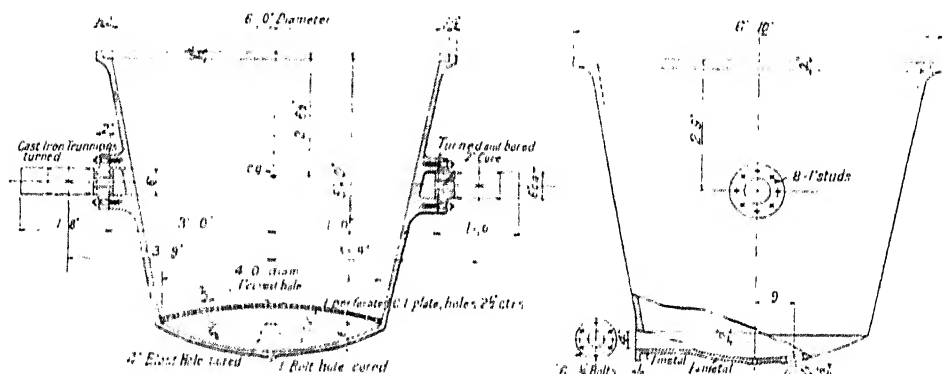
material to a dry-mixer, from which it passes to a pug mill with  $\frac{3}{4}$ -inch holes in the end, whence the mixture issues as semi-solid rods of that diameter. These, falling into a conical revolving trommel, are broken up into pieces of



Front view.

Side elevation (sectional).

Figs. 65 and 66.—General arrangement Carmichael-Bradford Pots.



Section.

Side elevation (part sectional).

Figs. 67 and 68. Carmichael-Bradford Pots.

marble size, and delivered on to a drying pan-conveyor, which, driven at 1 foot per minute through a drying furnace, 10 feet long, so as to dry and "set" the granules, drops them into the boot of an elevator, which takes them to a storage-bin provided with shoots over the converter pots.

The converters are of 4 tons capacity, 6 feet in diameter, and 5 feet high, of the form shown in Figs. 67 and 68, made of cast iron, with separate cast-iron trunnions bolted to the body, and turned by means of a worm- and hand-wheel. The blast enters through a 4-inch hole under the perforated false bottom. Blast for a plant treating 150 tons per day is furnished by two Baker blowers, driven by a 50 H.P. engine.

Figs. 65 and 66 are reproduced here\* because they exemplify the method adopted at Broken Hill of arranging charge bins as well as the hoods and pipes for carrying off the fumes.

By proper control of the blast supplied, the fumes carried off by the hoods over the pots can be made to carry as high as 14 per cent. of  $\text{SO}_2$ , but it is preferred to keep them at about 10 per cent., in which case there is sufficient oxygen present to oxidise the  $\text{SO}_2$  to  $\text{SO}_3$ . The gases are led over into pots to lead chambers of 40,000 cubic feet capacity. One ton of sulphide mixture with 14 per cent. sulphur yields nearly 12 cwt. of chamber acid, or 7 cwt. of strong acid. The new works at Port Pirie are to treat per annum 35,000 tons of ore, and to turn out 10,000 tons of acid for the manufacture of superphosphate. The desulphurisation effected is usually from about 14 per cent. S down to 4 per cent., or, say, 70 per cent. of the sulphur present, the loss of lead is negligible, and that of silver is only 1 per cent.

After repeated experiments it was found that lead concentrates (54 per cent. Pb) gave good results with an admixture of 25 per cent. gypsum, as did slime (21 per cent. Pb) with 30 per cent. gypsum, the increase in weight of the converted product over that of the original lead-bearing material being from 10 to 11 per cent. The following table shows the composition of the original materials and converted product in each case :

TABLE XIV.—ANALYSES—CARMICHAEL BRADFORD PROCESS.

	CONCENTRATES			SLIMES		
	Original.	Gypsum.	Converted product	Original	Gypsum	Converted product
Pb, . . . . .	54.0	..	48.9	18.0	..	16.1
Zn, . . . . .	9.4	..	7.0	16.6	..	14.0
Fe, . . . . .	1.9	..	1.8	6.0	..	5.4
Mn, . . . . .	0.9	..	0.8	2.5	..	2.25
S, . . . . .	14.6	..	3.9	13.1	..	3.6
Insoluble, . . . .	19.2	18.9	21.75	38.5	18.9	39.8
CaO, . . . . .	..	24.8	5.8	2.1	24.8	8.6
Al <sub>2</sub> O <sub>3</sub> , . . . . .	..	4.5	1.0	3.2	4.5	4.1
Fe <sub>2</sub> O <sub>3</sub> , . . . . .	..	0.5	Undet.	..	0.5	Undet.
SO <sub>3</sub> and H <sub>2</sub> O, also O,	..	51.3	8.15	..	51.3	6.15
Total, . . . . .	100.00	100.0	100.00	100.00	100.0	100.00
Average ozs. per ton,	24 ozs.	..	22 ozs.	19.2 ozs.	..	12.5 ozs.

\* From *E. and M. J.*, Oct. 28, 1905, p. 779.



It is, however, now found most convenient to treat a mixture of slimes and concentrates in the proportion of 3 parts of the former to 1 of the latter, and only 25 per cent. of gypsum is added.

*Cost of the Carmichael-Bradford Process.*—Under Broken Hill conditions, with unskilled labour at 7s. 6d. for eight hours, gypsum at 10s. per ton, and coal at 33s. 9d. per ton, the cost per ton of roasting by this process is estimated as follows :—

	s.	d.
0.25 ton of gypsum, . . . . .	2	6
Dehydrating and granulating gypsum, . . . . .	2	0
Drying mixture of ore and gypsum, . . . . .	0	6
Converting, . . . . .	1	0
Spalling sintered material, . . . . .	0	6
0.01 ton of coal, . . . . .	0	4
Total, . . . . .	6	10

The present practice at Port Pirie\* is somewhat different to that above described, being simpler, and may be shortly described as follows :—

The ore, together with from 10 to 20 per cent. crushed gypsum and a certain amount of crushed ironstone to assist in forming slag, is delivered to a mixer, where also it is sprayed with 4 to 5 per cent. of moisture, and the mixed material is taken by trucks or a conveyor to the converters, which are now made rectangular in section at the top, 6 feet long by 4 feet wide, and measure only 16 inches deep to the grate bars (movable), below which is a hopper bottom and a suction pipe to the gas main. The grate bars are first covered with a layer of infusible ironstone to protect them, and the charge is dumped into the converter through a wooden griddle with  $\frac{1}{2}$ -inch bars and  $1\frac{1}{2}$ -inch square spaces, through which it is scraped, in order that it may fall lightly without packing. When the converter is full the griddle is removed, and the charge is covered with about 10 lbs. of straw or wood shavings; the air valve is opened that connects the bottom of the converter with the suction main, from which air is being exhausted by ordinary rotary blowers, and the straw is ignited, giving heat enough to ignite the charge.

The best results are obtained by a suction of about  $2\frac{1}{2}$  inches of mercury ( $\approx 1\frac{1}{4}$  lbs. per square inch) at the commencement of the blow, falling to about  $\frac{5}{8}$  inch towards the end of same. During the progress of the blow the charge becomes more porous, and exposes less resistance to the passage of the blast, so that the volume of air which passes through increases gradually; when acid is to be made it is necessary to control the suction by means of valves, in order to avoid impoverishment of the gases, but not otherwise. Each charge takes 45 minutes to one hour to burn, sinter, and become cold, at the end of which time the converter is turned upon its trunnions by means of gearing, and the sintered cake falls from 6 to 7 feet upon the floor, the fall generally sufficing to break it into pieces of suitable size for the blast furnace. The modified plant is a great improvement on that originally described, and the present output is 1,500 tons per week.

**The Savelsberg Process.**† In this, the simplest of all the lime-roasting processes, fine granulated limestone to the amount of 15 or 20 per cent. by

\* *Australian Mining and Engineering Review*, Dec. 5, 1908, p. 77.

† W. R. Ingalls, *E. and M. J.*, Dec. 9, 1905; Wm. Hutchings, *E. and M. J.*, Oct. 21, 1905, p. 726.

weight is mixed with the lead ore, containing sufficient silica to form a slag, and the mixture is charged gradually in layers into a pot containing a small quantity of glowing fuel. The limestone appears to serve two distinct functions, one mechanical, in moderating and regulating the oxidation, and the other chemical. By virtue of its infusibility, it prevents premature fusion, and allows the charge to remain permeable to the blast, while at the same time reducing the temperature generated by the oxidation of the lead sulphide through its own decomposition into caustic lime and carbon dioxide, a reaction which is, of course, endothermic. The caustic lime becomes converted into calcium sulphate by absorption of  $\text{SO}_2$  and oxygen, and this again at a later stage of the process is in the lower part of the pot decomposed by silica, the  $\text{SO}_2$  liberated serving to facilitate the oxidation of lead sulphide still remaining in the upper part of the pot. As in all pot-roasting processes it is advisable to wet the charge mixture in order to prevent the escape of dust, to obviate

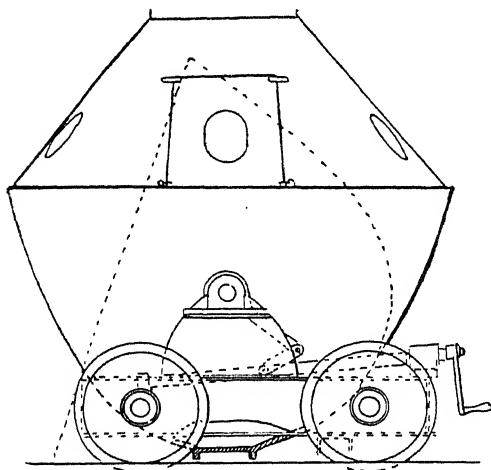


Fig. 69.—Savelsberg Converter.

the separation of the various ingredients in the charge, and to facilitate the production of sulphuric acid.

The Savelsberg converters are of hemispherical form, 8 feet in diameter, and weigh about 26 cwts. each, the capacity being 8 tons of ore mixture. Unlike the pots used in a majority of Huntington-Heberlein installations, which are fixed in position, or movable only by a travelling crane, they are set upon trucks as shown in Fig. 69.

The pressure of blast during the filling of the pot is kept low, only 4 to 8 inches of water (say  $2\frac{3}{4}$  to  $4\frac{1}{2}$  ozs.), while, after the pot is filled, it is allowed to rise to 20 to 24 inches of water ( $11\frac{1}{2}$  to  $13\frac{1}{2}$  ozs.). The blowing operation lasts 18 hours, and the agglomerated ore contains from 2 to 3 per cent. sulphur. One man per shift (in Westphalia of 12 hours) attends to the working of two pots, apart, of course, from the labour of filling and of breaking up the roasted mass.

As in all the pot-roasting processes the breaking up of the roasted charge

presents some difficulty. At Ramsbeck the sintered cake is tipped on to a vertical iron bar, which breaks it usually into several pieces.

*Loss of Lead and Silver.*—At Ramsbeck, in roasting by the Savelsberg process, the loss of lead is said to be not over 1 per cent., and that of silver nil. This is on a galena ore containing 60 to 78 per cent. of lead, mixed with 10 per cent. of spathic iron ore, 10 per cent. of siliceous silver ore, and 19 per cent. of limestone, so that the resulting ore mixture contains about 15 per cent. sulphur and 11 per cent. silica.

According to some experiments by Hofman,\* the loss of lead, which rose to  $2\frac{1}{2}$  per cent. with only 15 per cent. limestone on the charge, was reduced practically to nothing when the limestone was increased to 26 per cent. The loss of silver, on the other hand, which, with 5 per cent. limestone, was only  $1\frac{1}{2}$  per cent. of the total present, increased steadily with successive additions of limestone until with 25 per cent. it reached 6 per cent. of the total present. With lead ores low in silver, however, the increased loss in silver is more than compensated by the diminished loss of lead.

According to some experiments by the author, even rich galenas with 78 to 80 per cent. lead can be partially roasted by this process, provided the ore and limestone are broken to a  $\frac{1}{4}$ -inch mesh or less, and thoroughly mixed, and that care is taken not to let the blast pressure exceed 4 ozs. at first. The blowing lasts 18 to 20 hours, and much poking is required to equalise the exit of gases and fumes over the surface of the charge. The blast pressure rises to 10 or even 12 ozs. at the very end of the operation. The elimination of sulphur is imperfect, from 13 per cent. in the original down to 7·8 per cent. in the product, but most of this is in the condition of sulphate. The tendency to sinter is, however, considerable, and some metallic lead is apt to be produced through reaction in the lower and hotter part of the pot. On the whole, it is decidedly more advantageous that the charge for blast roasting should be made to contain a lower percentage of lead, by admixture of non-lead ores, or by other means appropriate to the conditions.

*Pot-roasting without Lime.*—At some modern American plants a system of pot-roasting is employed for mixed ores without the addition of lime, but, of course, the percentage of lead in the ore-charge is much lower than in those treated by the lime-roasting processes, and the preliminary partial roast of the original Huntington-Heberlein process is retained.

Austin† describes the *modus operandi*, and illustrates the plant employed by a section which is reproduced in Fig. 70. The pot is 8 feet 6 inches in diameter, with detached hood communicating with the dust flue, and is provided with a cast-iron perforated plate having  $\frac{3}{8}$ -inch holes, under which is a baffle plate which spreads the blast.

After spreading a barrow-load of ashes, 1 or 2 tons of hot roasted ore are added, and then the remaining 8 or 9 tons of damp partially roasted ore, which contains Pb 7 to 10 per cent., SiO<sub>2</sub> 30 per cent., and S 12 to 14 per cent. The blast pressure commences at only  $\frac{1}{2}$  oz., and is raised gradually to only 4 ozs.

At Murray‡ (Utah), this process is carried out on a large scale in a battery

\* *Trans. Am. Inst. M.E.*, Jan. 1907, p. 37.

† *Min. and Sci. Press*, xc., Oct. 27, 1906, p. 511.

‡ W. R. Ingalls, *E. and M. J.*, lxxxiv., Sept. 28, 1907, p. 575; and *Min. Ind.*, 1907, vol. xvi., p. 656.

of 25 pots with spherical bottoms, each 9 feet in diameter, and taking a 9-ton charge about every 12 hours. The pots are of cast iron, with long heavy trunnions set in open bearings, so that they can be lifted bodily by a crane and taken to a tipping floor. A steel hood over the pot leads, by means of a telescoping section of pipe, to an elevated circular steel flue with V-shaped bottom. The charge is part roasted ore (8 to 12 per cent. S, FeO 20 per cent., SiO<sub>2</sub> 10 per cent.), but the greater part is raw ore (18 to 25 per cent. S, FeO 20 per cent., SiO<sub>2</sub> 40 per cent.); the lead is kept down to 20 per cent., and the zinc to 10 per cent. In charging, 1½ inches of hot coal ashes are spread over the grate, followed by 1 ton of hot Godfrey "special" roast with 8 per cent. sulphur, which in its turn is followed by 1 ton of hot, dry Godfrey "common" roast with 12 per cent. of sulphur; finally, 7 tons of a damped

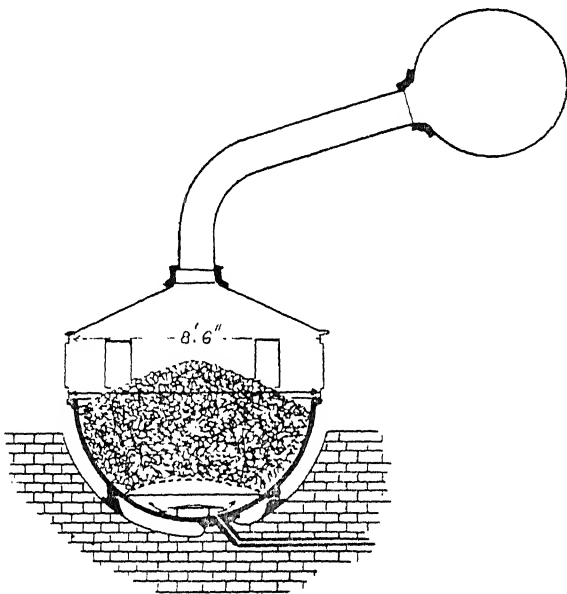


Fig. 70.—Blast Roasting-pot (American Type).

mixture of raw and roasted ore, carrying about 20 per cent. S and previously mixed by hand in an old pot. As soon as the surface is levelled, the full blast at 12 ozs. pressure is at once turned on; the operation lasts from 8 to 14 hours (averaging 12), and the sulphur is reduced to 4 per cent. When finished, the pot is raised by a crane, tilted so as to pour off the non-agglomerated, partially roasted fines on top, which go to the next charge, and then carried to the breaking-floor, where it is dropped from a height upon a platform of steel rails to break it. Large lumps are picked up by the crane and dropped again, smaller lumps are fed into a 10 × 20 Blake crusher, which delivers it to a double wire-rope conveyor with steel buckets, discharging into large railroad cars.

The total amount of labour required by this large plant, treating 450 tons

per day, is only ten men per shift—viz., six on the pot-floor and four on the breaking-floor.

The methods and the plant employed in the new works at *Garfield* (Utah) are almost identical, except that the ores there are different, being mostly copper-bearing. The pots are in two sections, bolted together; the charge is 1 ton hot McDougall-roasted concentrates with 12 per cent. S, and then 6 tons of wetted McDougall-roasted concentrates, and is roasted down to 6 per cent. sulphur in 8 to 10 hours.

**Blast-roasting in Grate Furnaces.**—At *Cerro de Pasco* (Peru)\* blast-roasting in a thin layer is carried out in so-called "grate ovens," which consist simply of a grating pivoted in the centre with a movable hood above and a stone-built bin below. Fine ore and flue-dust are delivered from different hoppers to a conveying-belt, upon which they are sprayed with water and delivered to a screw conveyor, in which the material is mixed, and from the end of which an elevator raises the mixture to hoppers. The grating has an area of 124 square feet, has holes tapered from above downwards, and is suspended upon a central axis, so that it may be rotated to dump the charge when finished. Crushed limestone is first spread over the grating, and then a layer of the cold wet ore and flue-dust mixture. This is ignited by means of a layer of wood chips scattered over the top and hot embers from a travelling hearth on wheels, which runs the length of the ovens. Blast at a pressure of 5 ozs. is turned on through two pipes passing through the hood, and the charge sinters from above downwards; the operation normally takes only one hour. The charge contains 10 to 14 per cent. copper and 20 per cent. sulphur, which is reduced to about 4 per cent. in the sintered product. Each grate oven treats about 30 tons per 24 hours, but, owing to channelling, the roasting is often very irregular and the sintering poor.

An experimental plant of blast-roasters at the works of the *United States Smelter* (Utah) has a battery of low rectangular grate furnaces with fixed sides and sliding front and back doors. The blast is admitted below the grate, and when finished the front and back doors are opened, and the cake of sintered charge is pushed out by a ram, like the coke from a horizontal coke oven. No details are available as to the working of this appliance, but it ought to give satisfactory results in practice.

At the *Bingham Junction* works of the U.S. Smelting and Refining Coy. a battery of 26 rectangular furnaces has been recently put in, each 6 feet  $\times$  6 feet  $\times$  3 feet deep, provided with a hood overhead and a perforated grate through which enters a blast under 9 ozs. pressure. A starting charge of coal and pyrites concentrate is first spread on the grate to a depth of 2 inches, and after this is well ignited the ordinary ore mixture is added in 2- or 3-inch layers, each being allowed to ignite before the next is added. The total charge is 8 tons, and the time required for sintering is 10 to 12 hours. The solid cake is forced out upon a steel "boat" by means of a ram as coke is forced out of a by-product oven, and is then lifted by a travelling crane and dropped upon a steel floor to break it, the process being completed by dropping a cast-iron weight upon the lumps, and then putting them through a large Blake crusher.

**The Dwight and Lloyd Sintering Process.**†—This latest development of blast-roasting processes appears destined to have a great future. It aims

\* *M.S.P.*, Jan. 30, 1909, p. 195.

† *E. and M.J.*, March 28, 1908, p. 650.

at carrying on continuously, by means of appropriate mechanical appliances, the reactions which are brought into play intermittently in ordinary pot-roasting. In ordinary pot-roasting, after the blast has been turned on, some individual layer of the charge is always being actively oxidised and undergoing the sintering operation. As the roast proceeds, however, in addition to the layers near the top of the pot, which are idly awaiting their turn to enter the reaction zone as the latter travels steadily upwards, there are others which have already undergone the reactions; after which they are not merely inert, but to some extent actively obstructive, since they interfere with the even upward passage of the blast on its way to the reaction zone. Any method, therefore, of eliminating these idle and dead layers, and of subjecting to the action of the blast only such a thickness of ore as can enter the reaction zone practically all at once, will naturally give a capacity much greater in proportion to the size of the apparatus employed than can be secured by means of pots.

One form of apparatus designed by Messrs. Dwight and Lloyd is shown in Fig. 71. It consists in the main of a slowly revolving drum resting on rollers, the circumference of which is formed of a series of cast-iron perforated gratings. Inside the upper quadrant of the circle is a fixed suction box, connected with a suitable suction fan, and the revolving rims make a fairly air-tight joint with this. The material to be sintered is fed from a hopper upon the revolving grate in a thin layer, making a layer about 4 inches thick, which, as the circumference revolves, is passed under a series of gas or oil jets, or a charcoal brazier, to ignite its upper surface, after which, as the drum continues to revolve, the heating and sintering action gradually progress inwards, being completed for a layer of the thickness mentioned in about twenty minutes. Discharging is effected automatically by means of a grizzly of upward-curved bars, the points of which are fixed close to the revolving grates forming nearly a tangent thereto, by means of which the sintered material is neatly stripped off, forming porous and coherent masses in ideal condition for the furnace.

At the *Maurer* (N.J.) works of the American Smelting and Refining Co. a machine of this type treated a 50 per cent. galena concentrate without addition of lime at the rate of 30 tons per 24 hours, using a 4-oz. vacuum, the power required to maintain which was 12 H.P. The sulphur content of the ore was reduced to 3.4 per cent., and the gases filtered through bags showed a loss in metal of less than 0.5 per cent.

The most modern type of continuous machine is the so-called Straight Line machine.\* This consists essentially of a series of "pallets," which are pushed into contact like the pans of a pan-conveyor, but not attached to each other by chains or otherwise. Each pallet is 30 inches wide by 18 inches long, with a perforated bottom planed on its lower side to make a good joint when passing over the suction box, and supported by four wheels at all other points in the circuit. Each pallet successively is pushed up into position, receives a 4-inch layer of charge from a feed-hopper, passes under the ignition furnace, and then over the suction box, during its passage over which roasting and agglomeration is completed, so that the roasted cake is dropped from the pan on to guides, which carry it into a hopper. The suction box

\* Hofman, International Congress of Chemistry, London, 1909, report in *Mining Journal*, June 12, 1909, p. 728; also *Priv. Comm.*, A. S. Dwight, Aug. 9, 1909.

is 12 feet 6 inches long by 30 inches wide, so that the effective hearth area of the appliance (which is the measure of its roasting capacity) is  $31\frac{1}{4}$  square feet. The ignition box employed is a small reverberatory furnace built of

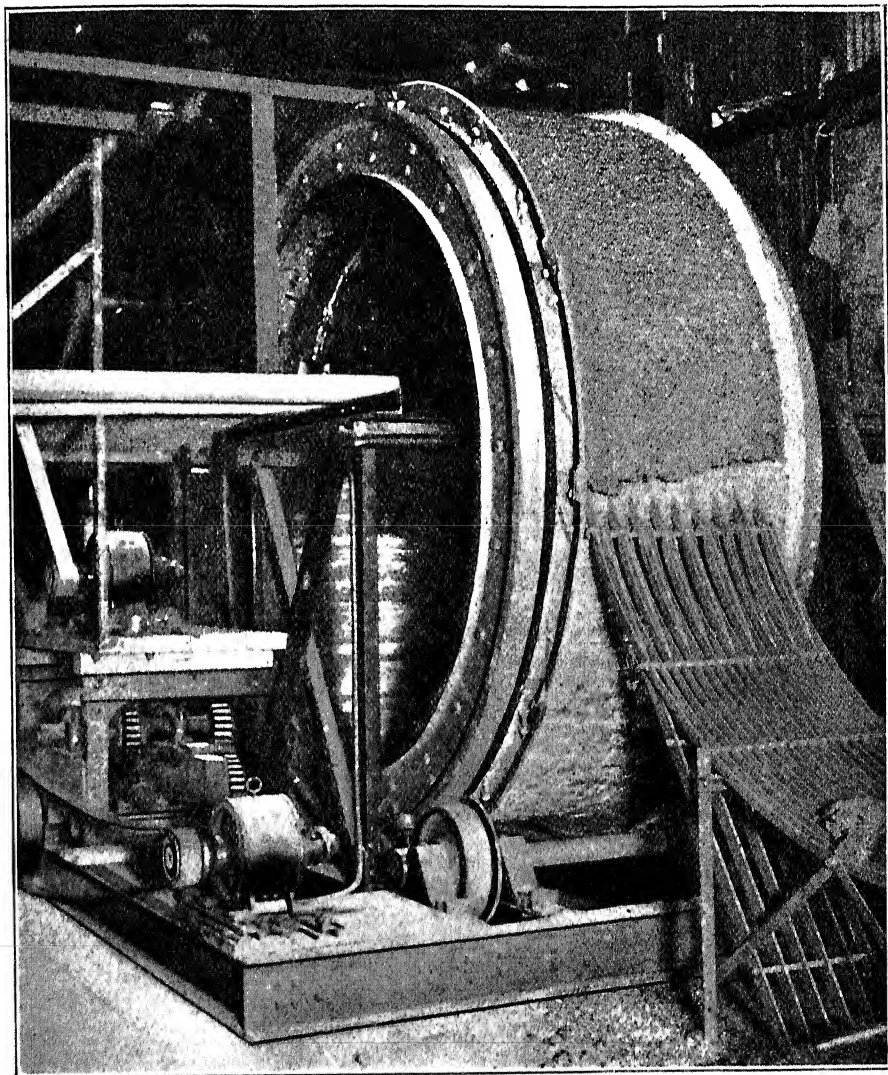


Fig. 71.—Dwight-Lloyd Sintering Machine—Drum Type.

fire-tiles, having a grate area 30 inches by 10 inches, and burning 500 lbs. of coal per 24 hours; a line of gas- or oil-jets can, however, be employed. The total weight of the machine without accessories is 16 tons. The per-

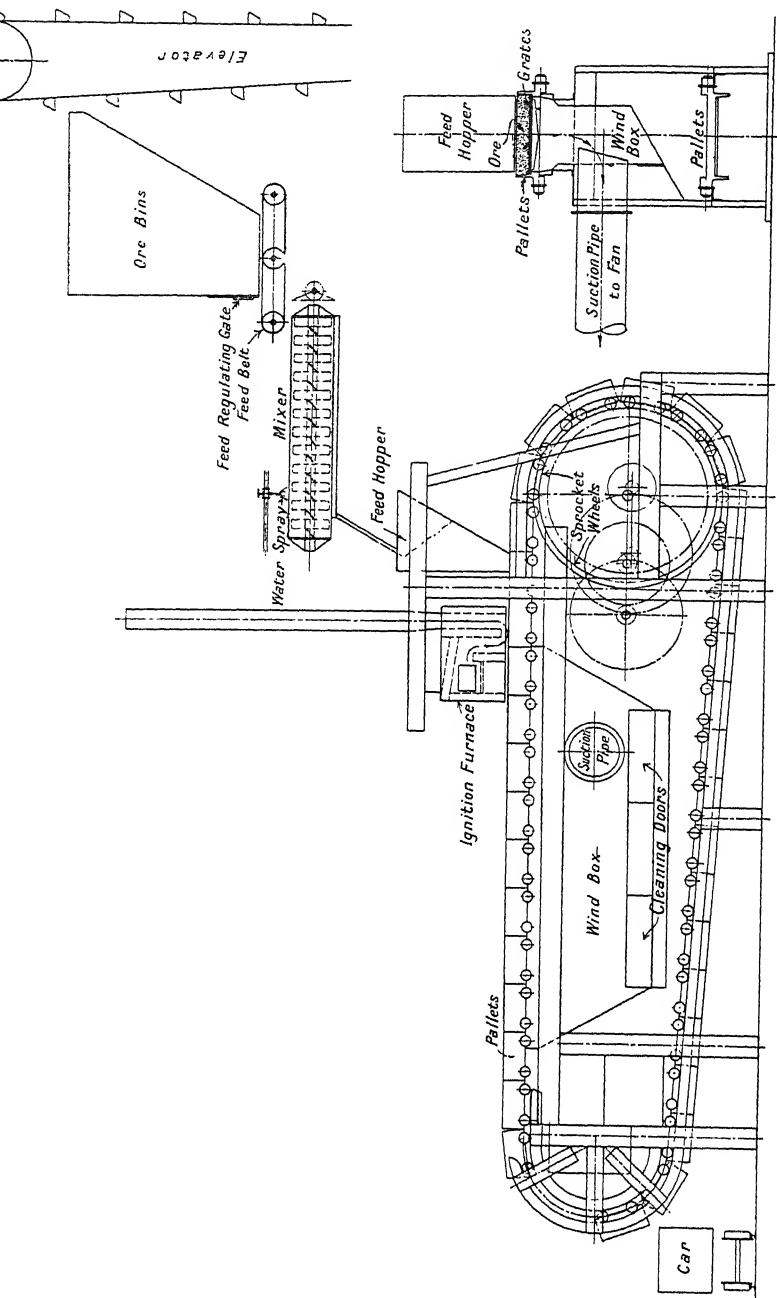


Fig. 71A. — Dwight-Lloyd Sintering Plant. "Straight Line" (Pallet) Type.



forated bottom of the pallets is formed of replaceable cast-iron grate-bars of herring-bone pattern having quarter-inch spaces. Cleaning of these grates is seldom necessary, but there is ample time for this on the return journey, as also for replacing grate-bars, or even pallets, when required, without stopping the machine. The mode of operation is easily seen by reference to Fig. 71A; each pallet successively is pushed by the teeth of the large pair of sprocket wheels and lifted from the lower track on to the upper, along which it continues to be pushed by the succeeding pallets until eventually it is pushed off at the discharge end and guided by the circular guides back to the foot of the sprocket wheels. The speed of travel can be varied from 5 to 15 inches per minute according to the material; with coppery ores carrying from 9 to 18 per cent. sulphur, the capacity may be only 30 tons per day, and with heavy pyritic ores a good deal less, while with leady concentrates as much as 75 tons per day may be put over the machine and agglomerated. The tonnage roasted per sq. ft. of effective grate area thus varies from 0.6 up to 2.5 tons per day, which is very high compared with all other roasting appliances.

For best results it is desirable to keep the percentage of sulphur in the charge mixture between 9 and 18 per cent.; ores, therefore, which contain more than 18 per cent. are mixed with flue-dust, which increases the porosity of the mixture besides reducing the total percentage of sulphur present, and so facilitates roasting. In any case water is added to the mixture, so that it will "ball" readily in the hand (equal to about 8 to 10 per cent. moisture), and so prevent its falling through the openings in the grates. An ordinary leadworks' mixture of fine ores through a quarter-inch screen, and containing  $\text{SiO}_2$  25 per cent., Fe 25 per cent., Pb 25 per cent., and S 15 per cent., is passed through one of these machines at the rate of 55 tons per day, with a coal consumption of 500 lbs. per day for ignition, and a power consumption of 1 H.P. for the sintering machine itself, 6 H.P. for the mixer, elevator, &c., and 20 to 25 H.P. for the exhaust fan, which handles some 6,000 cubic feet of air per minute under a vacuum of 4 to 7 ozs., the temperature of the gases being  $100^\circ$  to  $150^\circ$  C. The sulphur in the sintered product varies from  $3\frac{1}{2}$  to 7 per cent., and the total operating cost varies from 50 cents (2s. 1d.) up to 75 cents (3s. 1½d) per ton in single unit plants, being less in larger plants. The volatilisation loss is said to be lower than in pot-roasting plants, owing to the shorter time during which each particle of ore is exposed to a high temperature.

It is important that the composition of the charge should be kept as uniform as possible, since any considerable variation results in an unsatisfactory product.

At the *Ohio and Colorado Smelting and Refining Co.* (Salida, Colo.) a machine of this type, 30 feet long, treats 50 tons of ore per 24 hours with a rate of travel of 8 inches per minute (equals about 19 minutes for the completion of the roasting and agglomerating process upon each 4-inch layer of ore). One man can look after several machines, and the total cost of treatment, including transport of the ore 200 feet, and of the sintered product the same distance, is said to be 50 cents (2s. 1d.) per ton.

## CHAPTER VII.

## BLAST-FURNACE SMELTING—PRINCIPLES.

**Introductory.**—When treating of the blast-furnace smelting of lead ores, it will be clearly recognised that in a majority of instances the pure galena ores, which alone are suitable for the Scotch hearth and reverberatory processes, play but a comparatively minor part, and form but a small proportion of the blast-furnace charge, the bulk of which is usually composed of more or less complex ores of the precious metals, particularly of silver, for which the lead plays the part simply of a collector or vehicle. Even in Germany, the classical home of blast-furnace smelting, the charges employed contain on an average not more than 30 per cent. of lead; and in American practice nowadays the average is generally not over 10 to 12 per cent., the limit perhaps having been reached in the smelting of gold ores at *Freemantle* and *Dapto*, in Australia, where the percentage of lead on the charge was as low as 8 to 9 per cent.

Roasting is an indispensable preliminary to the blast-furnace smelting of galena ores, but the importance of a perfect roast has greatly diminished since the introduction of the modern system of fast driving, with its enormous volume of air supplied through tuyeres of large area. In the German system of smelting with a small number of tuyeres of small size, giving a powerfully reducing effect with slow smelting, the whole of the sulphur present goes to form matte. By using a larger volume of blast, however, it is possible to burn off a considerable portion of the sulphur present, and, therefore, an imperfectly roasted charge, or even a mixture of raw and roasted ore, may be employed without much increasing the production of matte.\*

Although *roast-reduction*, therefore, is the typical blast-furnace process, at the present time both *reaction* and *precipitation* are in a minor degree exemplified at different centres.

*Roast-reduction* predominates in the (ordinary) case where the lead ores have been reduced pretty thoroughly to an oxidised condition by natural agencies or by a thorough roast. Lead is reduced to metal chiefly by the agencies of carbonic oxide and of glowing carbon; and sulphur chiefly forms an iron-lead matte, except a small portion expelled as  $\text{SO}_2$ , and another small portion carried into the slag as zinc, lead, iron, and barium sulphides. These conditions prevail at most of the great lead-smelting plants in the world.

*Precipitation* predominates in the exceptional case where the lead ores are charged in an unaltered sulphide condition together with oxidised iron ores and other fluxes in high furnaces, a small volume of air only being employed, involving a very slow rate of smelting, together with a very high fuel consumption. The atmosphere in the furnace is powerfully reducing, and

\* The volatilisation loss, however, is in this case much increased.

lead sulphide is acted upon by metallic iron reduced from the fluxes, producing a small quantity of iron-lead matte. The typical example of this condition of affairs is to be found at *Clausthal*, but it may be presumed that precipitation always comes into play to a greater or less extent when smelting is slow, and when the other conditions are favourable to a powerfully reducing action. The practice in the United States, during the eighties, undoubtedly involved more or less "precipitation."

*Reaction* invariably comes into play to a greater or less extent whenever sulphide and partially oxidised ores of lead are charged into the furnace together, and may be said to predominate when the ores are imperfectly roasted, and when flue-dust or similar oxidised material is briquetted together with fine lead concentrates or other sulphide fines. Reaction is also aimed at in the exceptional practice which consists in charging mixed, raw, and roasted ores in a comparatively low column into a blast furnace, the atmosphere of which is made oxidising by the expedient of supplying it with a larger volume of air than can be burnt up by the coke at the given pressure. Lead is chiefly liberated by the reaction of its sulphate and oxide upon unaltered sulphide, and the sulphur present in the charge is chiefly expelled as  $\text{SO}_2$ . This practice in its typical form was exemplified at the *Block 14 Smelting Works, Port Adelaide*,\* but to some extent reaction, no doubt, comes into play wherever a large increase in furnace capacity has been aimed at by increasing the volume of the blast without very greatly increasing the height of the furnace. Whenever an increase of capacity obtained by this means without altering the composition of the charge is found to be unaccompanied by an increased matte fall, it is safe to reckon that some reaction effect has come into play, and this condition of affairs, though unusual, is occasionally reached at several large European and Australian lead smelting establishments.

Although, however, *precipitation* and *reaction* are always more or less concerned in blast-furnace lead smelting, they play a part which is, on the whole, distinctly subordinate to the more universal *roast-reduction* process. The principles of blast-furnace smelting will, therefore, be discussed chiefly on the *reduction* basis, with only incidental reference to such exceptional conditions.

**Objects of Blast-Furnace Smelting.** The charge consists, as a rule, of both raw and roasted ores mixed in convenient proportions, a majority of the ores valuable chiefly for their silver contents being smelted in a raw condition. It may contain, besides oxide, sulphate and sulphide (sometimes silicate) of lead, silver and gold, the former combined as sulphide or native, also oxides, sulphides and sulphates of iron, zinc, and copper, arsenides and antimonides (arseniates and antimoniates), free silica and silicates of iron, alumina, lime and magnesia, barium and calcium sulphates, and also, in the raw ores, carbonates of lime, magnesia, iron and zinc, fluorspar, and other gangue minerals.

It is mixed with fuel and with appropriate fluxes and smelted, the object being to produce :—

1. A "work-lead" or "base-bullion," which shall contain as much as possible of the lead, silver, and gold present.

\* Now, however, no longer in operation.

2. A "matte," which shall contain as much as possible of the copper, together with the rest of the lead, silver, and gold.

The remaining ingredients are to be got rid of by slagging or by volatilisation, but when much arsenic is present an additional product is formed in the shape of speiss, which complicates the extraction.

**Requirements of the Ore Charge.**—The ore charged should contain for good work sufficient lead to yield at least 8 to 10 per cent. (and preferably 12 per cent. or upwards) on the whole weight of charge including fluxes; and the proportion of silver should not be more than 3, or at most 4, ozs. for each per cent. of lead shown by dry assay. The other constituents of the charge—silica, iron, lime, &c.—should have been, as far as possible, balanced by mixing various classes of ore, so as to produce a "good" slag with the minimum amount of barren fluxes.

**Requirements of the Slag.**—The slag produced must be good, economically as well as metallurgically. To be good *metallurgically* a slag must fulfil the following conditions:—

1. Low formation point, provided that it is high enough for properly carrying out the furnace reactions.

2. High degree of fluidity when molten.

3. Low specific gravity.

4. Low contents of lead and silver.

1. The melting points of ordinary lead slags may vary between 1,030° C. (Iles) and 1,273° (Balling), being lowest when the composition approaches to a bisilicate of iron with about 10 per cent. lime, and highest as it approaches to that of a pure monosilicate of iron or mixed sesquisilicates of iron and lime, mixtures of monosilicates of iron and lime occupying an intermediate position.

2. When producing lead bullion with 200 to 300 ozs. silver, a good slag should contain, according to Eilers,\* not over  $\frac{3}{4}$  per cent. lead (by dry assay) and  $\frac{1}{2}$  oz. silver per ton (by ordinary crucible assay uncorrected). These figures, however, are often somewhat exceeded as averages on the large scale, even in thoroughly well-managed plants.

3. The separability from matte and speiss depends partly on the *density* of the slag (3·3 to 3·6) and partly on its *viscosity*. It is known in a general way that siliceous slags are less perfectly fluid than those of a basic character, while both *zinc* and *magnesia* tend to render slags pasty, but no experimental investigations to determine viscosity appear to have been made; at all events, the author is unaware of any.

To be good *economically* a slag, besides satisfying the above conditions, must permit of smelting as much as possible of the ore on which there is the largest margin of profit, with the minimum of costly flux. Not only the cost of flux delivered at the works has to be borne in mind, but the cost of smelting it, for every pound of flux added to the furnace charge means a pound of ore left out. It costs about as much to smelt a ton of flux as a ton of ore, therefore great attention should be directed to utilise as far as possible the various kinds of ore available, so as to employ as little barren flux as possible, and that of the cheapest kind.

**Fluxes.**—The fluxes used in the lead blast furnace are:—

1. *Silica* in the form of siliceous silver ores, used only for basic

\* *E. and M. J.*, April 9, 1881.

ferruginous ores. This is not usually spoken of as "flux," that name being reserved for basic fluxing materials, and (in Colorado) for the lead ores which are added to siliceous mixed ore charges.

2. *Iron Ore* (also *manganese ore*).—This is a favourite flux for siliceous ores whenever obtainable, more especially because, when derived from the gozzans of ore deposits, it usually carries more or less silver and gold, and sometimes lead. Its cost alone precludes its more universal use for simple fluxing purposes when not carrying any "values." All iron ores contain more or less silica, which, besides diminishing their fluxing power, forms an unnecessary amount of slag in the furnace. This is almost always objectionable, not only on account of the cost of smelting worthless material, but also because of the lessened furnace capacity for ore, and of the extra losses of lead and silver in this additional slag.

When smelting very zincy ores which are not too siliceous, however, it is sometimes preferable to use a cheap siliceous iron flux rather than the more expensive high-grade iron ore, as the extra amount of slag formed assists in carrying off zinc.

*Hammer scale* and *puddle cinder* are still employed at Mechernich, and, perhaps, at one or two other places, but are far too expensive for use at most great smelting centres.

It has been stated that *magnetite* is unsatisfactory as a fluxing material, being likely to give rise to furnace irregularities, but if not too fine and the column of charge not too low, according to the author's experience, magnetite gives quite as good results as hematite. With a low charge column, however, fine magnetite might give trouble through getting into the slag and matte unfused.

It is impossible too strongly to impress upon the novice that in the ordinary lead blast furnace a deficiency of iron lies at the root of most of the difficulties and irregularities in furnace manipulation, except those caused by zinc, and that the troubles caused by an excess of base (especially of iron) are much less to be feared than those due to an excess of silica. Lime is usually far cheaper than iron, both absolutely and relatively—*i.e.*, taking into account its superior satisfying power, but lime in a slag cannot be run up with impunity beyond 28 to 30 per cent. as a maximum, while with zincy ores it must be kept within far lower and somewhat narrow limits, as will be seen later.

It is often necessary to economise iron-fluxing ores on account of their expense, or the cost of roasting, if the iron flux has to be prepared on the spot from pyritous ores, but a pretty safe rule for the inexperienced metallurgist to follow is "when in doubt or in trouble increase the iron ore on the charge."

3. *Limestone*.—This is the flux most widely used, the purest kind being the best for furnace work. Generally, however, the best obtainable is more or less siliceous or argillaceous, and frequently also dolomitic. Siliceous and argillaceous impurities are objectionable in exactly the same way as when found in iron ores—*viz.*, because they bring about the formation of an unnecessary amount of slag. Magnesia, theoretically, should be more efficacious than lime for neutralising silica, on account of its lower molecular weight; but, unfortunately, the silicates of magnesia are so much less fluid at temperatures above their formation point than the corresponding silicates of lime as to

more than offset this advantage when used in large proportion. The use of highly dolomitic limestones is, therefore, objectionable, especially when the charge is high in zinc, though, as will be seen later, it appears to be advantageous with charges high in alumina or in alkalies.

The proportion in which lime may be used varies greatly. In the practice of thirty years ago (chiefly owing to the abundance of iron ores and the ferruginous excess carried by many carbonate ores) limestone was added to the charge only in small proportion, and the slags contained but little lime. Nowadays, on the other hand, the price of iron flux being usually higher and the cost of roasting greater, a large proportion of limestone is advantageously employed, ranging, when the zinc present is small in amount, up to a maximum of 30 per cent. CaO in the slag produced. The use of lime enables us to decrease the combined lead in slags (*i.e.*, to decompose lead silicate), and also by lowering their specific gravity, allows of a better settling-out of the superseded shots of matte and lead, so producing cleaner slags than could possibly be arrived at when iron was the only base employed. Moreover, as already stated, a given weight of lime will neutralise more silica than will the same weight of ferrous oxide, so that a given quantity of silica in the charge produces a smaller weight of slag with smaller losses of lead and silver. Limestone, therefore, irrespective of its lower first cost, is in most cases the most advantageous flux.

*Slags* from the same or a distinct operation are frequently added as flux with one or more of the following objects:—(1) To recover lead or silver contained in them; (2) to open up a fine dense charge and prevent it from clogging the blast; (3) to add some basic or acid constituent in which the charge is deficient; and (4) simply to promote fusion of the charge.

However well a furnace is running, some slag will always be produced containing shots of matte or speiss, or otherwise too rich to throw away (so-called "foul slag"); this will generally amount to from 5 to 10 per cent. of the ore charge. Slag having been already once melted is much easier to remelt than an ore mixture comprising the same constituents, therefore the addition of slag to a refractory mixture will assist it to melt more easily.

It is a remarkable fact that up to a certain point the addition of slag to the ore-charge so promotes fusion and increases the total tonnage smelted that the additional fuel required to melt it is hardly noticeable. This may be partly explained by considering that the heat lost in furnace gases, jacket water, &c., is not increased in proportion to the increased tonnage put through, and also that the first fusible slag which melts somewhat high up in the furnace acts as a solvent for the other slag-forming ingredients, individually of more refractory character, through which it percolates in its downward course. The fact remains that when smelting a somewhat refractory ore-mixture with no slag on the charge, or only a very small quantity, the addition of, say, another 200 lbs. slag to a 2,000-lb. charge, provided that slag is fusible, will generally not call for any increase in the normal fuel charge, and may actually increase the tonnage of ore put through. Beyond a certain point, however, determined by experience, further additions of even a fusible slag call for a proportionate increase of fuel to melt them, and seriously affect the tonnage capacity of the furnace for ore.

**Use of Burnt Lime instead of Limestone.**—According to Bell,\*

\* *Manufacture of Iron and Steel*, London, 1884, p. 60.

burnt lime cannot be economically substituted for limestone, even in an iron blast furnace, on account of the absorption of moisture and  $\text{CO}_2$  in the upper part of the furnace, which would have to be expelled lower down. Burnt lime was, however, substituted for limestone in one of the Ormesby furnaces at Middlesbrough\* with very satisfactory results, owing to the saving of fuel ordinarily required to expel  $\text{CO}_2$ , which forms 40 per cent. by weight of the raw limestone. Besides a saving of something like 7 per cent. of the coke, better reduction was obtained, as shown by the less loss of iron in slags. The advantages should be even more evident in the case of lead furnaces, for the descent of the burnt lime from the charging floor to the reduction zone is too quick to admit of much absorption of  $\text{CO}_2$ , while the reduction and fusion zones of a lead furnace are so restricted that there is a very real loss of temperature and fuel in decomposing limestone in the furnace. According to Le Chatelier, calcium carbonate requires a temperature of  $812^\circ$  for its decomposition, and it is pretty certain that this temperature would not be reached until the limestone was nearly down to the top of the jackets—i.e., close to the smelting zone, where all the heat of the fuel is required for iron reduction and slag formation.

According to Bretherton,† burnt lime was used instead of limestone at the *American* smelter (Leadville, Colo.) from 1885 to 1893, with the result that the furnace ran faster, the slag was hotter and cleaner, and fewer wall accretions were formed. Other metallurgists who have tried burnt lime have found difficulties due to its friability and pulverulent condition when partially slaked, and it would seem that it is essential for the lime to be (a) freshly burnt, and (b) in large lumps of, say, 5 to 6 inches in longest dimension. When these conditions can be satisfied the advantages of burnt lime would seem to be as follows:—(1) No crushing or shodging required; (2) saving of fuel owing to absence of reaction between the expelled  $\text{CO}_2$  and carbon; (3) increase of furnace capacity for ore proportional to the difference of weight; (4) no loss of heat in smelting zone, therefore hotter and cleaner slags; (5) less total quantity of gas produced in furnace, therefore less volatilisation loss; and (6) possible direct action of the porous lime in upper part of the shaft upon sulphides of Zn and Pb volatilised from smelting zone, with formation of  $\text{CaS}$  and diminution in the amount of wall accretions formed. It is to be noted that in some few cases the saving in freight caused by the less weight might pay for the cost of burning. It would probably be difficult, however, to realise any advantage from the employment of ready-burnt lime, except in a dry climate, and where the limekilns were in close proximity to the furnaces.

#### FUELS EMPLOYED.

The fuels best suited to blast-furnace treatment are **coke** and **charcoal** either used together or separately in various proportions. **Coal** and **wood** have been tried as partly replacing coke, and the result of experiments in their use will be detailed, but, generally speaking, it may be stated that the drawbacks to their use are—First, that their coking in the upper part of the smelting zone absorbs heat just where it is most wanted, while tending

\* *Cochrane, Journ. I. and S. Inst.*, May, 1898; also *E. and M. J.*, vol. lxx., June 25, 1898, p. 760.

† *E. and M. J.*, March 5, 1893.

to heat the charge near the top, which should be kept cool. Second, that the sudden changes in volume, due to the progress of the carbonisation (especially of wood), and the increase in the quantity of gases given off, tend to increase the volatilisation of lead and generally to cause irregularities in the furnace working.

**Coke and Charcoal.**—Comparing coke with charcoal it is to be noted that, on account of the tendency of all lead compounds to sinter together at a low temperature, comparatively little reduction can be effected by CO, and actual contact of molten drops of oxides and silicates with masses of glowing fuel has principally to be depended upon. Since CO, therefore, plays but a small part in the economy of the furnace, the fuel should be burned before the tuyeres as far as possible to CO<sub>2</sub>, so as to give out as much heat as possible just where it is most needed; at the same time, the portion of the charge above the smelting zone should be kept as cool as possible, so as to prevent the CO<sub>2</sub> from being converted into CO by taking up carbon according to the reaction  $\text{CO}_2 + \text{C} = 2\text{CO}$ , with consequent waste of fuel and loss of heat.

Coke fulfils these conditions much better than charcoal, for, owing to its greater density and less porosity, much more CO<sub>2</sub> is produced in burning it before the tuyeres than is the case with charcoal; while its higher igniting point renders it much easier to keep the smelting zone concentrated into a narrow belt and prevent the heat from rising (with consequent waste of fuel, increased volatilisation, decreased porosity of the charge, and other disadvantages which follow a rise of heat), a condition specially liable to occur when charcoal is exclusively used.

Another advantage possessed by coke over charcoal is its greater strength, which enables it to resist the weight of a much higher column of charge without crushing; and this is important, especially when dealing with mixtures poor in lead and rich in iron requiring much lime flux, for it is impossible to properly smelt such mixtures except in a high furnace.

Charcoal labours under the disadvantages of breaking up in handling, and of crumbling or decrepitating in the furnace. Crushed charcoal is not only worthless as fuel, but makes foul slags and increases the amount of flue-dust. On the other hand, charcoal has less ash than coke, keeps the charge more open (on account of its greater bulk), and is more porous; hence weight for weight a fuel charge of mixed coke and charcoal will smelt more ore in a given time in a low furnace than one composed wholly of coke. The fuel consumption when using charcoal alone is, however, excessive, owing to the crushing which takes place, and which often reaches 30 to 40 per cent., and for this reason charcoal cannot be used at all except with a low charge column; moreover, it must be used freshly burnt, since it deteriorates rapidly when stored.

The common practice among American lead smelters in the eighties was to use a mixture of coke and charcoal, by which, to some extent, the combined advantages of strength and porosity were realised, the coke supporting the charge, while the charcoal, igniting first, burnt away quickly leaving hollow spaces among the coke for the blast to penetrate. With modern high furnaces of ample size and slags rich in lime, however, the use of charcoal has been entirely abandoned in favour of coke; since, against a possible trifling decrease in furnace capacity (offset by the increased blast pressure



permitted by the higher charge-column), and a small increase in the quantity of slag formed, the use of coke offers, in addition to great economy, the advantages of cleaner slags and smaller amount of flue-dust. The coke used in Western America is usually soft and ashy, published analyses being no criterion of the latter point. Thus of cokes which are supposed to contain from 12 to 15 per cent. ash, like most of those from Southern Colorado and New Mexico, the author has frequently found truckloads with as much as 21 to 23 per cent. ash, and has had whole train loads averaging  $19\frac{1}{2}$  per cent. This has to be considered in the consumption.

**Anthracite** of goose-egg size has been substituted by Ropp\* for part of the coke charge of a furnace, with the result that when 60 per cent. of the coke was so replaced the furnace capacity was reduced by one-third, the working remaining normal in other respects, the furnace top being cool and the crucible hot, while the slags were clean and matte and speiss both low in lead. With charges containing  $7\frac{1}{2}$  per cent. zinc, smaller wall accretions seemed to be formed than when using coke alone.

At *Freemantle* (W. A.) † one-fifth of the normal fuel charge consisting of English coke with 5 per cent. ash was replaced by a corresponding amount of Collie (non-bituminous) coal with 7 per cent. ash. The furnaces ran faster and fewer wall accretions were formed, while, the coal costing only one-third as much as the coke replaced, there was a considerable saving.

**Bituminous Coal** of assorted sizes from pea to lump has been used by Neill‡ to replace charcoal and part of the coke in a mixed fuel charge of coke and charcoal. With 27 per cent. of bituminous coal the crucible kept more open, the lead and slag appeared hotter, the tuyeres brighter, and the slag was cleaner than when coke alone was used. The smoke was thicker, but there was no more flue-dust on account of the furnace top being cooler. The charges settled more evenly, and fewer zinc accretions formed than with coke alone.

Bituminous coal was also employed successfully by Austin§ at the *Germania* Works, Salt Lake City, for eighteen months, but was abandoned owing to the arsenical and coppery ores smelted, as it was found to increase the amount of "top crusts" formed on the lead. At these works the fuel charge of 14 per cent. coke and 2 per cent. charcoal, which cost \$1.33 per ton of ore, could be replaced by 9 per cent. coke, 2 per cent. charcoal, and 6 per cent. nut coal, costing \$1.07 per ton of ore smelted. Austin notes that coal produces a very explosive mixture at the feed floor, and that it tends to make the charge denser, so increasing the danger of this explosive mixture coming back into the blast main. He says—(1) That coal can only be profitably used when the saving in price over coke is considerable; (2) that it should always be combined with charcoal to counteract its density; (3) that the total fuel charge should be from 1 to 2 per cent. more than with coke alone; (4) that the feeding should be even and carefully watched; and (5) that it should not be used with ores carrying much arsenic and copper.

**Gas and Oil Fuels** have been tried in the blast furnace, and have successfully replaced as much as 30 per cent. of the coke charge. The substitution would seldom be a financial success, and from what will be seen about the reducing effects of solid carbon and of CO or other gaseous-reducing agent,

\* *Trans. A.I.M.E.*, vol. xx., p. 171.

† *Trans. A.I.M.E.*, vol. xx., p. 165.

+ L. Pitblado, *Priv. Com.*

§ *E. and M. J.*, Dec. 15, 1894.

taking into account, moreover, the important mechanical function performed by a porous solid fuel in keeping the charge open, it is plain that under no circumstances could the replacement of solid fuel in a blast furnace be carried much further.

**Amount of Fuel Required.**—The quantity of fuel required per ton of ore smelted is least :—

1. With ores rich in lead and, therefore, producing less slag.
2. With ores having a fusible gangue and comparatively free from zinc, magnesia, and other refractory ingredients.
3. With coarse ore, which permits free penetration of the blast and quick running of the furnace.
4. At the level of the sea, where the volume of blast required is less.
5. In summer, when the blast, being already at a temperature say 30° to 50° above that of winter, the fuel required is fractionally less.
6. In dry weather generally, when there is least moisture to be evaporated from the fuel, ore, and fluxes.

It is, of course, greatest under diametrically opposite conditions—*i.e.*, poor ores with refractory gangue in a fine condition, smelted at high altitudes in wet weather. The extreme variations in practice are with coke from 12 to 22 per cent., with charcoal 20 to 28 per cent., and, in exceptional cases, even over 30 per cent. As an average, it may be reckoned that using a fair quality of coke with ordinary ore mixtures from 11 to 14 per cent. of coke will be required on the charge.

The way in which fuel consumption is affected by altitude calls for some attention. A plausible explanation often advanced is that where the atmospheric pressure is low the products of combustion have to be discharged in a more expanded condition, which means that they carry away a greater aggregate amount of heat: the total heat units generated by the complete combustion of a given amount of carbon being always the same, when a larger number are carried away in the expanded furnace gases, fewer are left inside the furnace, and more coke must, therefore, be burnt to keep up the heat of the smelting zone. This explanation is fallacious, for although it is true that the expansion of the gases at a higher altitude requires slightly more fuel, the increase on this account is only fractional, and quite insufficient to account for the observed increase necessary. The real reason consists in the fact that the temperature generated by the combustion of a given body is not absolute or invariable, but depends strictly upon the number of units consumed in a given time in a space of given size. Thus, a candle burning in compressed air not only burns faster than in air at the ordinary pressure, but the temperature of its flame is higher because of the increased intensity of combustion. If, on the other hand, the candle be burned in a partial vacuum, where a smaller number of molecules of oxygen can reach it in a given time, the temperature of the flame gets lower and lower till it finally goes out; deduction of the heat units lost by radiation and convection from the total number generated by the combustion not leaving sufficient to keep up the required temperature. Seeing, therefore, that the pressure inside the furnace produced by the blast is only a certain constant amount above the atmospheric pressure of the locality, it is evident that the intensity of combustion and, therefore, the temperature produced by the burning of a given amount of fuel will be less the higher the altitude; and excess fuel must be burnt in order to keep up the required temperature. This explanation

further accounts for the observed fact that furnace capacity also falls off rapidly with increased altitude.

**“Tuyere Ratio” and “Tuyere Efficiency.”**—It is unfortunate that the habit of running furnaces from one general blast main, coupled with the use of the cheap and convenient revolving blower, the actual “slip” of which is seldom determined, have rendered it impossible hitherto to ascertain, with much approach to accuracy, the volume of air which is supplied to any furnace; and on this account it is not usually possible to properly investigate the character of the work done inside the furnace under given conditions.

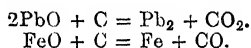
In the absence of satisfactory data as to the volume of air supplied, it is convenient, for the sake of comparing furnaces of different sizes, to make use of certain relations between the tonnage of ore smelted, or rather, strictly speaking, the tonnage of fuel consumed, the sectional area of the furnace at the tuyere level, and the total area of the tuyeres. The number of tons of material smelted in twenty-four hours per square foot of sectional area may be called, as first suggested by Lang, the “hearth activity,” or “hearth efficiency” of the furnace; this will be affected not only by the volume of air supplied, but also by its pressure.

As no data are obtainable in most cases with regard to the volume of air supplied, it may be assumed to be, for any given height of charge-column, roughly proportional to the total area of the tuyeres. The number of square inches of tuyere area provided for every square foot of effective sectional area between the tuyeres may be conveniently termed the “tuyere ratio” of the furnace. With any given size of furnace and fixed “tuyere ratio,” the quantity of ore smelted in twenty-four hours, or, in other words, the quantity of fuel consumed, will be directly proportional to the pressure of blast. The number of tons smelted in twenty-four hours per square inch of tuyere area may therefore be called the “tuyere efficiency” of the furnace. The “hearth efficiency” of any given furnace will thus be the product of the volume factor, “tuyere ratio,” into the pressure factor, “tuyere efficiency.” These factors are worked out for the furnaces, details of which are given in a subsequent table, as well as for the furnaces smelting silver ores without lead, described in the volume on the *Metallurgy of Silver*.

#### REACTIONS OF THE LEAD BLAST FURNACE.

**Reduction and Reducing Agents.**—The principal reducing agents in the lead blast furnace under ordinary conditions are carbon and carbonic oxide; under exceptional conditions sulphur, iron, and metallic sulphides also act as reducing agents.

Carbon acts directly on metallic oxides at the bright red heat of the smelting zone, the product being carbonic acid with easily reducible oxides, and carbonic oxide with those which are more refractory.

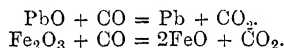


The carbon of the fuel burnt before the tuyeres at a white heat is converted chiefly into  $\text{CO}_2$ , as proved by Schertel,\* but a certain proportion is burnt directly to  $\text{CO}$ . Of the  $\text{CO}_2$  formed a portion is reduced to  $\text{CO}$  by

\* Wagner's *Jahresberichte*, 1880, p. 188.

contact with glowing fuel in the centre of the furnace. A few inches above the tuyeres, therefore, the furnace gases contain a large proportion of CO.

**Carbonic Oxide.**—The carbonic oxide present in the furnace gases immediately above the tuyeres cannot, however, exert any action till it rises to a point where the temperature is less than 1,200° C., because it can only reduce by self-oxidation to CO<sub>2</sub>, a gas which begins to dissociate at that temperature. Hence the reducing action of carbonic oxide commences some little distance above the tuyeres. It acts by production of CO<sub>2</sub> as follows:—



The amount of CO<sub>2</sub> in the furnace gases is also increased by the decomposition of carbonates contained in the ore and fluxes, which takes place above the smelting zone. The decomposition temperatures of the ordinary carbonates found in smelting charges are as follows: \*—PbCO<sub>3</sub>, below 200° C.; MgCO<sub>3</sub>, 650° C.; FeCO<sub>3</sub>, 800° C.; CaCO<sub>3</sub>, 850° C.†

According to Iles,‡ the temperature of the escaping gases from lead blast furnaces in good work may range up to 334° F., but the average is only 214° F. (101° C.). Schertel analysed nineteen samples of gases from Freiberg blast furnaces, and found on an average three times as much CO<sub>2</sub> as CO, besides small quantities of hydrogen and of marsh gas. Some other analyses are given in Chapter x.

**Conditions Favourable to Reduction.**—The burning of carbon to CO or CO<sub>2</sub> is largely influenced by the shape of the furnace and by the area of the tuyeres. A distinct contraction of the furnace at the tuyere zone (bosh) is accompanied by a much more rapid descent of charge there, and, consequently, by the accumulation of a thicker layer of glowing fuel between the liquid slag and the unmelted charge. The CO<sub>2</sub> produced before the tuyeres in its ascent has to pass through this layer of coke, and so becomes converted into CO before reaching the unmelted charge. During its slow passage upwards through the constantly increasing width of charge, the CO has ample opportunity to become oxidised to CO<sub>2</sub> by absorbing oxygen from the ore which it reduces. In the case of a furnace with vertical sides, however, the layer of glowing coke is thinner and its temperature is lower, because the heat is less concentrated; the CO<sub>2</sub> formed, therefore, largely remains as such.

Similarly, small tuyeres deliver streams of air, which are rapidly and completely absorbed by the glowing coke in their path to form CO, while the quantity delivered by a large tuyere is so great that the oxygen is largely in excess of the available carbon at any point close in front of the tuyere; hence combustion takes place almost entirely with the formation of CO<sub>2</sub>, which in its passage upwards through the thin bed of glowing coke has no time to be converted into CO.

Another factor which influences the reducing action of the furnace is the blast pressure. A powerful blast forces its way to the centre of the furnace, where both any oxygen unconsumed on the way and the CO<sub>2</sub> molecules formed by the combustion are surrounded by an excess of glowing carbon, and are rapidly converted into CO, which then rises through the charge, exerting

\* Hofman, *op. cit.*, p. 206.

† According to Le Chatelier, 812° C.

‡ *S. of M. Q.*, vol. xvii., p. 19.

its reducing effect on the metallic oxides. A weak blast, on the contrary, penetrates the charge with difficulty. Such portion as reaches the centre is, of course, converted into CO, but a large part is burnt to  $\text{CO}_2$  in the immediate neighbourhood of the tuyeres, and rises through the charge, together with any oxygen which has escaped burning without being reduced by the fuel. In this way, although there is reduction in the centre, oxidising effects predominate all round the shaft.

Very small charges again, which bring about an intimate admixture of fuel and ore in the furnace, increase the reducing action; while the use of large charges results in fuel and ore reaching the smelting zone successively instead of together, and to that extent is not so favourable to reduction. Similarly, a fine-ore charge hinders the draught, and causes the furnace gases to become more thoroughly disseminated through the ore column, so exercising a thorough reducing action; whereas with a coarse charge a considerable portion of the blast can escape through open channels without exercising any reducing influence on the bulk of the ore. Moreover, an extreme degree of coarseness and openness in the charge has a bad effect in causing the heat to rise too high in the furnace, whereby not only is there a great waste of heat in the escaping gases, but the fuel is consumed at so great a height above the tuyeres as to seriously lower the temperature in the smelting zone, and give rise to imperfect separation, growth of accretions, and other troubles.

The rapidity with which coke can be burnt in a furnace of given area with tuyeres of given size, and, therefore, the rate of smelting, depend upon the pressure of the blast; and any increase in this demands a corresponding increase in the height of the column of charge, in order to avoid waste of heat at the top. High pressure, therefore, and high furnaces mean rapid driving, especially when combined with a large tuyere area.

The following conditions are favourable to a powerful reducing effect:—

1. High column of charge.
2. Contracted tuyere zone (boshed furnaces expanding upwards).
3. Small tuyeres.
4. Small volume of blast at a high pressure.
5. Small charges of moderately fine ore.

Exactly the opposite conditions viz., low column of charge, straight-sided furnaces, and a large number of tuyeres of large area supplying a great volume of air at low pressure to large charges of coarse material—produce an atmosphere which may be neutral or even oxidising in its effects.

**Composition of the Ore Charge.** The composition of various lead ores before roasting has been given in Table X., and Table XV. gives the composition of certain roasted lead ores as well as of some natural “carbonate” ores added for comparison.

**Reactions of the Ore Components.** Some raw ore being added to the roasted ore, the chemical composition of the ore charge is often very complex. The different substances present behave as follows:—

Lead oxide in powder or in porous lumps is partly reduced in the upper part of the furnace direct to metallic lead by carbonic oxide. Slagged or sintered masses are, however, impermeable to the furnace gases, and their reduction is only effected in the smelting zone by direct contact of the drops of molten oxide with glowing fuel,

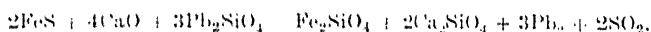
TABLE XV.—ANALYSES OF ORES FOR BLAST-FURNACE SMELTING.

Carbonates.											
Roasted Sulphides.											
Reference,	Freiberg, Saxony.	Mechernich, Rhenish Prussia.	Rodna, Transyl- vania.	Goslar, Lower Harz.	Pueblo, Colorado.	Mine La Motte, Mo., "Galena."	Mine La Motte, Mo., "Sulphides."	Eureka, Nevada.	Red Mountain, Colorado.	Sourcez, Laurium.	Camareza, Laurium.
	1.	2.	3.	4.	5.	6.	7.	8	9.	10.	11.
PbO,	22.00	66.85	58.44	10.02	7.91	77.40	21.86	35.65	18.40	10.50	6.95
Ag,	0.13	0.01	0.06	0.01	0.06	...	...	0.09	0.44	0.031	0.0152
CuO,	0.30	0.17	0.02	0.93	3.22	...	7.04	0.15	...	...	0.12
ZnO,	20.00	0.19	1.03	23.64	0.98	und.	und.	2.37	1.23	14.95	3.00
FeO,	...	...	...	...	...	...	...	...	...	2.78	26.75
Fe <sub>2</sub> O <sub>3</sub> ,	33.30	0.78	34.37	13.62	80.39	4.31	34.37	34.39	17.12	...	...
MnO,	...	0.10	...	...	0.93	1.11	4.69	0.13	0.59	2.89	7.32
NiO and CoO,	...	0.08	0.03	...	tr.	...	...	...	...	...	...
Sb,	...	...	0.01	...	0.86	...	...	0.25	...	...	...
As <sub>2</sub> O <sub>3</sub> ,	1.10	...	0.04	...	...	...	...	6.34	...	0.10	7.50
Al <sub>2</sub> O <sub>3</sub> ,	1.80	4.24	0.23	...	...	und.	und.	0.64	6.08	0.25	10.70
CaO,	2.00	1.18	tr.	7.16	3.22	2.25	3.28	1.14	1.70	19.42	2.20
MgO,	0.50	...	tr.	...	...	...	1.45	0.41	...	0.25	0.10
BaSO <sub>4</sub> ,	...	...	...	20.87	...	...	...	...	...	...	...
SiO <sub>2</sub> ,	17.40	22.77	0.80	11.00	3.21	2.80	7.10	2.95	4.63	3.40	8.64
SO <sub>3</sub> ,	tr.	und.	2.25	8.62	...	...	...	4.18	3.87	...	...
S,	3.60	0.60	2.72	0.38	1.01	7.12	8.31	...	2.39	0.22	1.30
CO <sub>2</sub> ,	...	...	...	...	...	...	...	...	6.75	...	16.00
H <sub>2</sub> O,	...	...	...	...	...	...	...	10.90	...	...	...
CaF <sub>2</sub> ,	...	...	...	...	...	...	...	...	...	25.96	...
	102.13	96.97	99.99	96.25	101.79	...	...	99.59	100.20	...	...
Ag, oz. per ton,	42.50	3.4	19.9	4.25	20.0	...	...	30.74	143.4	10.5	5.25
Au, oz. per ton,	...	...	...	...	...	...	...	1.96	0.23	...	...

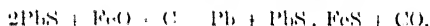
References.—1. *Oesterreichisches Jahrbuch*, vol. xvi., p. 397. 2. *Berg- und Hüttenm. Zeitung*, 1875, p. 129. 3. *Oesterr. Jahrb.*, vol. xxix., p. 27. 4. Schnabel, *Handbuch der Metallhüttenkunde*, vol. 1., pp. 329 and 389. 5. Hofman, *Metallurgy of Lead*, p. 260. 6 and 7. Kemp, *School of Mines Quarterly*, vol. ix., p. 216. 8. Curtis, *Mon. U.S. Geol. Survey*, vol. vii., 1884, p. 60. 9. Kadzie, *Trans. A.I.M.E.*, vol. xvi., p. 570. 9 and 10. Author, *Private Notes*, 1904.

Lead silicate is unaffected by CO, and even by contact with glowing carbon, except in the presence of a more stable base; basic lead silicates only give up a part of their lead, leaving an acid silicate. This is, however, readily decomposed by FeO, and at a much higher temperature by CaO also, and the PbO set free is reduced to metal by hot coke. The ferrous oxide acts equally well whether in a free condition, reduced from ferric oxide in raw or roasted ore, or combined with silica, from which combination lime will set it free in the smelting zone. In ordinary lead furnaces more or less metallic iron is reduced from its oxide in the hottest part of the furnace in contact with glowing coke, and this iron acts as a powerful reducing agent for the liberation of lead from both oxide and sulphide.

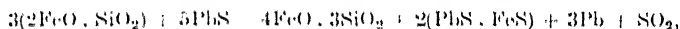
In connection with an excess of lime or other base, the sulphides of iron, calcium and barium, may act as reducing agents. These sulphides may be formed by reduction of their respective sulphates or by the action of pyrites on the bases, and their mode of action is as follows :—



Lead sulphide is acted upon in the hottest part of the furnace by metallic iron, or by mixtures of ferrous silicates with carbon, and reduced to metal with formation of ferrous sulphide; but the reaction is never complete, some lead always remaining as sulphide in the matte—



Percy gives\* an equation, according to which the presence of carbon is not necessary to the reduction of PbS by means of iron silicate, thus,



the monosilicate becoming a sesquisilicate which has no further action upon PbS. At a higher temperature more lead can be expelled from the matte by the action of fresh iron, so that the proportion of lead in the matte varies almost entirely with the temperature. Berthier stated that lime also reduced lead from its sulphide, but, according to the experiments of Jordan,† there is no action between the two bodies at any temperature, whether in presence of carbon or not.

A certain proportion of lead is always liberated from lead sulphide above the smelting zone by reaction with oxide and sulphate, according to the characteristic equations given in Chap. i. In some cases this reaction may become of the very highest importance, and nearly the whole of the sulphur present in the charge, even when amounting to 8 or 10 per cent., may be expelled as SO<sub>2</sub>, almost without any formation of matte. The conditions upon which this reaction depends have been already touched upon in this chapter, and will be further discussed in connection with the description of *Block 14 Works* (Port Adelaide).‡

Finally, lead sulphide may be reduced to metal by the direct action of zinc vapours (themselves arising from the reduction of oxide in contact with carbon), producing zinc sulphide.

It is to be noted that most of the lead lost by volatilisation is in the condition of sulphide, as shown by the analyses of lead fume from blast furnaces

\* *Metallurgy of Lead*, p. 59.

† Quoted by Percy, *op. cit.*, p. 55.

‡ *v. Chap. xi.*

run with a cold top; and this is only natural when we consider that lead sulphide volatilises rapidly at a temperature of  $860^{\circ}\text{C}$ ., which is far below its melting point.

Lead sulphate behaves in several ways. Usually the largest part reacts upon lead and other sulphides above the smelting zone, liberating metallic lead and  $\text{SO}_2$ . Another portion is probably reduced to  $\text{PbS}$ , and a third portion, in contact with free silica or with acid silicates, is converted into lead silicate; the lead sulphide and silicate thus formed are acted upon as above described.

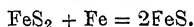
Ferrous carbonate is reduced to  $\text{FeO}$  with expulsion of  $\text{CO}_2$  at, according to Le Chatelier, a temperature of  $800^{\circ}\text{C}$ .

Ferric oxide and magnetic oxide are reduced by  $\text{CO}$  to  $\text{FeO}$ , which combines with silica to form the basis of a slag. In the smelting zone, part of the iron is still further reduced to metal, which then decomposes lead sulphide, as already explained.

Ferrous sulphate and basic ferric sulphate can hardly exist in large proportions in a roasted ore, but when present are decomposed to  $\text{Fe}_2\text{O}_3$  and  $\text{SO}_2$ .

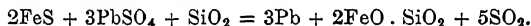
Ferrous sulphide melts at  $925^{\circ}\text{C}$ ., and in an ore charge chiefly runs through the furnace unaltered; a part of it, however, may undergo the double decomposition with lead silicate, already described on the preceding page.

Pyrites,  $\text{FeS}_2$ .—It has been sometimes supposed that pyrites, when added in the form of raw ore to the furnace charge, consumes an equivalent quantity of metallic iron, according to the equation—



Generally, however, it is reckoned with more reason that one atom of sulphur is given off in the upper part of the furnace,  $\text{FeS}_2$  thus becoming  $\text{FeS}$ . According to the experiments of Valentine,\* pyrites does not give off as much as one atom of sulphur when heated in a current of furnace gases ( $\text{N}$ ,  $\text{CO}$ ,  $\text{CO}_2$ ). After one hour's exposure to a temperature of  $678^{\circ}\text{C}$ . the residue still contained 37 per cent.  $\text{S}$ , showing that only a little over one-third had been volatilised; the greater portion of this, moreover—viz., 12 per cent. (23 per cent. of the total sulphur present)—was expelled in the first ten minutes. It seems probable that with charges rich in pyrites about three-sevenths of the  $\text{S}$  would be expelled at a temperature of  $500^{\circ}\text{C}$ ., or sufficiently below that at which  $\text{Fe}_2\text{O}_3$  begins to be reduced to metal, and that, therefore, at most only about one-eighth of the total contained sulphur has to be reckoned as consuming reduced iron to form matte.

According to Henrich,† pyrites may act directly upon lead sulphate, just as in the experiments of Percy it was found to act directly upon  $\text{PbO}$ . In the case described, so-called "carbonate" ores (consisting chiefly of anglesite) were smelted direct with pyrites containing silver, and very little matte was produced, most of the sulphur seeming to be burnt to  $\text{SO}_2$ . Allowing for the volatilisation of sulphur from the pyrites in the upper part of the furnace, the reaction may be represented as follows:—



In practice, it is found that with unroasted pyritic ores, apart from the mechanical condition of the charge, the amount of sulphur expelled is largely

\* Desulphurisation of iron ores, *Trans. A.I.M.E.*, vol. xviii., p. 78, &c.

† *E. and M. J.*, Sept. 22, 1893.



a matter of blast and of the speed at which the furnace is run; other conditions remaining the same, slow running generally diminishes, and fast driving increases, the proportion of matte produced from a charge with fixed percentage of sulphur.

Manganese oxides are reduced to MnO and enter the slag. Manganous oxide can replace ferrous oxide in all proportions; while the melting point of a slag is only slightly raised by the substitution of manganese for iron,\* its fluidity is so much increased that separation is improved, and much cleaner work can be done than with the ordinary iron slag.

Church,† in smelting carbonate concentrates at *Tombstone* (Ariz.) to a 314-oz. bullion, produced slag of the following composition:—

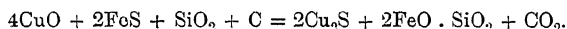
SiO <sub>2</sub>	FeO	MnO	CaO	MgO	Al <sub>2</sub> O <sub>3</sub>	Pb per cent.	Ag oz. per ton.
29·60	11·56	43·25	7·50	trace.	6·34	1·4	trace.

He notices one defect of slags rich in manganese, which is their slight power of taking up sulphur and other impurities, such as speiss (probably also barium and other sulphides). Pearce‡ also refers to the oxidising power of manganese on blende, giving a case of a slag produced in smelting silver-copper ores in a reverberatory furnace which contained SiO<sub>2</sub> 48 per cent., MnO 30 per cent., ZnO 12·5 per cent., while 3 per cent. of Mn also entered the matte. He points out that the presence of a large percentage of manganese in the slag ensures all the available sulphur going into the matte, and that so long as any manganese is present in the latter the copper is safe from slagging.

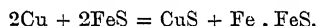
Chalcopyrite in raw ore loses about one-fourth of its sulphur, and melts down into a matte; other copper sulphides enter the matte unchanged.

**Cuprous and Cupric Oxides.**—The affinity of copper for sulphur being greater than that of any other metal, it passes into the matte, provided there be sufficient sulphur present; otherwise it will be reduced to metal and alloy with the lead. In order to get a perfect separation of copper in matte, however, the slags must not be too ferruginous, for with very ferruginous basic slags some of the sulphur is taken up by iron in the slag, and an equivalent proportion of copper goes into the lead. When running on coppery ores, therefore, slags fairly high in lime give cleaner lead.

The formation of cuprous sulphide from the oxide may take place in two ways. Either the oxide may be first reduced by CO and carbon to metal, which then acts upon ferrous sulphide, or a double decomposition may take place direct with the ferrous sulphide without previous reduction, according to the reaction:—



If the copper oxide is first reduced to metal, the reaction may be—



The metallic iron liberated and dissolved in the excess of FeS is at once disposed of by reaction upon lead sulphide or upon ferric oxide. Some have supposed the existence of a subsulphide of iron with the formula Fe<sub>3</sub>S, but no definite body of the kind has yet been prepared, and the principal reason

\* Hofman, *Trans. A.I.M.E.*, vol. xxix., p. 711.

† *Trans. A.I.M.E.*, vol. xv., p. 612.

‡ *Ibid.*, vol. xi., p. 59.

for assuming its existence is that the sulphur in lead mattes is frequently insufficient in quantity to satisfy the metals. On the other hand, metallic iron is sometimes found in mattes, and it is more reasonable to suppose that it exists dissolved in the hot matte,\* just as copper undoubtedly exists dissolved in the regulus known as "pimple metal" when hot, though it separates out on cooling.

Arseniates are partially decomposed by heat, especially in presence of pyrites, the arsenic being to a small extent volatilised as  $\text{As}_2\text{O}_3$ , but chiefly reduced to arsenides. In small proportion these arsenides are carried away, partly into the slag and partly into the matte (a trace only entering the work-lead); but when the proportion is at all high an iron speiss is formed, which contains notable amounts of Pb, Cu, and Ag, and from which these substances can only be recovered with difficulty and at some considerable expense. Such portion of the arsenic as may become volatilised carries with it into the fume both lead and silver, thus increasing the volatilisation losses of these metals. If, however, any great quantity of arsenic is present, most of it combines with metallic iron to form speisses of various compositions, the most common being  $\text{Fe}_5\text{As}_2$  and  $\text{Fe}_6\text{As}_3$ . Arsenic in a smelting charge, unless very small in amount, should be calculated to  $\text{Fe}_5\text{As}_2$ .

The formation of speiss, however, depends largely upon the reduction of iron. With the low charge column and slow speed common up to fifteen or twenty years ago, arsenic, whenever present in any quantity, invariably formed a speiss, owing to the excessive degree of reduction then practised. Nowadays, however, it is recognised that within limits it pays better to sacrifice reduction to tonnage capacity; the lead in slags is frequently a trifle higher than used to be considered good practice, but, on the other hand, the tonnage capacity is doubled, the fuel consumption is decreased, and arsenic is either carried into the matte or sent into the flues.

With highly arsenical ores it is advisable to make slags which are high in lime and somewhat more acid than the monosilicate type, even a sesquisilicate (No. 7 of Table XVII.) being sometimes used in order to keep the temperature of the pool of slag above the crucible as high as possible, and so prevent chilling of the speiss.

Antimony has less tendency than arsenic to form speiss, but quite as much to volatilise, carrying with it lead and silver; while it possesses the additional disadvantage of readily alloying with the work-lead, its separation from which introduces complications in the refining process. Antimonial speisses, when formed, always contain copper, and rarely much iron, unless arsenic be also present; they absorb more silver than arsenic speiss.

Antimoniates are reduced to antimonides, which then chiefly dissolve in or become alloyed with the lead, though a portion may enter the speiss or matte.

Antimony sulphide behaves in the main like lead sulphide, except that a much larger proportion is volatilised. The antimony reduced by metallic iron alloys with the lead, that portion, however, which is volatilised and converted into oxide carries away with it notable quantities of lead and silver.

Gold and Silver, in whatever condition present, are reduced to metal, and

\* See a recent paper by Fulton and Goodner, *Trans. A.I.M.E., Bull.* 24, Nov. 1908, pp. 80, 994.

alloy mainly with the lead. A smaller proportion of the silver, however, passes into the matte (chiefly owing to incomplete decomposition of silver sulphide), and a still smaller proportion into the speiss, if any is made. Traces of gold always accompany the silver, but the concentration of this metal in the lead is much more perfect than that of silver. Some silver is volatilised, but the amount is usually inconsiderable, unless haloid compounds of silver are present in the ore.

Alumina is generally unimportant in lead ores, but often forms an important constituent of the silver ores which are smelted together with lead ores. Its accommodating behaviour sometimes as an acid and sometimes as a base permits of regarding it as a practically neutral constituent of the slag, and as quite unimportant up to 5 or 7 per cent. Above that point its tendency to increase viscosity must be taken into account, although, even then, many metallurgists regard it as a neutral body in solution, which, therefore, does not require figuring as an essential part of the slag.\* In slags which are exceptionally high in lime, alumina tends to play the part of an acid, especially if the silica is also low. The behaviour of alumina in slags will be further referred to.

Lime decreases the specific gravity of a slag and raises its specific heat. Slags rich in lime, therefore, separate from matte more readily than do those rich in iron slags, and do not chill so quickly; on the other hand, they require a somewhat larger amount of fuel per unit of slag made. In proportions up to 30 per cent., lime lowers the melting point of ferruginous slags without increasing their viscosity, and thus facilitates the separation of the molten products, a result which, needless to say, is also much facilitated by the decreased specific gravity. To a limited extent lime at a high temperature can act directly on slag already formed, expelling  $\text{PbO}$  and  $\text{CuO}$  from their silicates, and assisting to produce a clean slag.

According to Schneider,† lime, besides acting as a strong base, diminishes the amount of matte formed in the furnace, and also the amount of lead going into the matte; this he explains by the formation of calcium sulphide, which is carried away by the slag in igneous fusion. The experiments recorded by Percy‡ would seem to contradict this view, which is, however, held by some experienced metallurgists, and it would seem to be undoubtedly correct at all events as regards slags formed in the iron blast furnace, although probably not as regards lead slags.

It is probable that, when iron ore or pyrites and calcareous material have both been employed in making up the charge for blast-roasting, the resulting product will contain much calcium ferrite,  $\text{CaO} \cdot \text{Fe}_2\text{O}_3$ , which, as shown by Hofman,§ commences to form at  $1,100^\circ \text{C}$ ., and melts at  $1,250^\circ \text{C}$ . This sub-

\* This is, however, not quite safe, as the following anecdote will show:—A very few years ago one of the principal smelting works in the S.W. of the United States was in the habit of receiving from Mexico an ore with about 12 to 15 per cent. alumina, which component, the ore being in other respects desirable, was not taken much note of in calculating furnace charges. One day, however, some of, apparently, the same ore came along containing about 40 per cent. alumina, and, without waiting for an analysis, it was used in making up a charge in about the usual proportion as if it had contained 13 per cent. alumina. As a result the slag had to be removed with a block and tackle instead of through the slag spout, for the furnace “froze solid.”

† *Trans. A. I. M. E.*, vol. xi., p. 58.

‡ *Metallurgy of Lead*, p. 55.

§ *Bull. A. I. M. E.*, Jan. 1909, pp. 69 and 70.

stance at its melting point would form a powerful solvent for siliceous material, and so much facilitate the formation of slag.

#### BLAST-FURNACE LEAD SMELTING—PRINCIPLES.

Calcium sulphate in the ore-charge with a low temperature, strongly reducing atmosphere, and basic slags, may be partly reduced to calcium sulphide, and removed in the slags as such. Under modern conditions, however, of high-pressure blast and high temperatures before the tuyeres,  $\text{SO}_2$  is expelled, and practically all of the lime slagged as silicate, the reaction commencing at about  $1,000^\circ \text{C.}$ , and being complete at  $1,250^\circ \text{C.}$ , as shown by Hofman.\*

Fluorspar is not of common occurrence in lead ores, but its extreme fluidity when molten renders it a valuable flux when it happens to occur as gangue matter. It possesses also the useful property of assisting in fluxing calcium and barium sulphates, causing them to be carried away in the slag in that form, and, according to Hahn,† it has a specially good effect in fluxing zinc sulphide. It is possible that a portion of the fluorine in fluorspar is expelled in the blast furnace (as, for example, is undoubtedly the case when blende ores containing fluorspar are roasted in reverberatory furnaces), but most of it enters the slag unaltered, and decreases its viscosity. It has been urged that too great a proportion of fluorspar in the charge might be deleterious in the same way as too high a proportion of alkalis may be harmful—namely, by lowering the formation temperature to such a point as to interfere with reduction. The melting point of fluorspar, however, is high, and it is clearly its comparatively perfect fluidity when molten rather than its low melting point which gives it value as a slag constituent.

The presence of fluorspar in ores may be detrimental in another way—namely, by causing determinations of silica and lime made in the ordinary way to be misleading. In the presence of several per cent. of fluorine the silica determination may be several per cents. too low; at the same time, the total percentage of lime present will be in excess by several per cents. of that available for fluxing silica. The cumulative effect of these two errors in slag calculation might be serious.

The slags at *Laurium* (Greece) contain several per cent. of fluorine, doubtless as calcium fluoride.

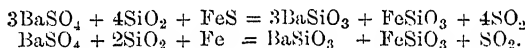
Magnesia in lead-smelting charges is calculated to lime, but in spite of its saturating power for silica, which exceeds that of lime in the proportion of 7 to 5, there was until recently much prejudice against the use of this base, on account of the fact that without raising the melting point of slags, it decreased their fluidity, causing them to be pasty or “mushy.” It seems to be generally recognised that magnesia in any considerable proportion is specially objectionable in presence of much zinc. Magnesia, however, in moderate proportions, is a good flux to use with barytic ores, as it keeps the specific gravity low without raising the melting point or decreasing the fluidity. One equivalent each of magnesia and baryta will practically take the place of two equivalents of lime, except for a slight increase in the density. Magnesia appears to be advantageously used together with high alumina,

\* *Bull. A.I.M.E.*, Jan. 1909, p. 63.

† *Trans. I.M.M.*, vol. viii., 1900, p. 269.

and also with alkalis,\* in which case its advantage lies in counteracting the tendency of the alkaline silicates to soften at too low a temperature.

**Barytes** is partly reduced to barium sulphide by contact with carbon, and partly through the action of FeS or of metallic iron in conjunction with silica at a high temperature, is converted into barium silicate, which, although itself comparatively infusible, forms with iron fusible double silicates. The reactions are—



It is probable, however, that a good deal of the baryta in lead slags exists in the condition of  $\text{BaSO}_4$ , for slags high in BaO leave a large residue of  $\text{BaSO}_4$  when decomposed with HCl. Such is certainly the case with the very basic slags of Julius and Sophien-hütte near Goslar (Harz) already referred to.

Under appropriate conditions BaS also enters the matte; thus the mattes made under exceptional circumstances in the Altai for the past forty years have carried from 15 to 20 per cent. barium, the slags being very siliceous, and carrying 5 to 15 per cent. BaO.

Baryta lowers the melting point of slags without reducing their fluidity; it notably increases, however, their density, and, except in small proportion, therefore, may be objectionable, except when MgO or ZnO are simultaneously present, when it acts as a powerful flux for these oxides, assisting them to slag, and much increasing the fluidity of the magnesia-zinc slag combination.

Potash and Soda, when present in an ore-mixture, usually enter the slag.† It has been pointed out by Carpenter‡ that the alkaline bases so lower the formation temperature of a slag as to render it viscous below its proper melting point, and he found that this tendency could only be counteracted by addition of magnesia.

Alkalies in a charge always increase the tendency of lead to pass into the slag, as was well exemplified in the case of the old slags made at Broken Hill, which contained about 2 per cent. alkalis, and averaged nearly 3 per cent. in combined lead, and by similar slags made at Freemantle. The reason for the high combined lead in alkaline slags may perhaps be that the formation temperature is too low to admit of thorough reduction.

Under exceptional circumstances, potash and soda may even enter the matte, as pointed out by Canby.§

Zinc oxide in the roasted ore chiefly goes into the slag, for, although simple silicates of zinc are very infusible, the basic double silicates of zinc and iron, and in a less degree those formed by zinc with lime and baryta, melt with comparative ease; the latter, however, are always somewhat pasty. If the proportion of zinc is more than about 8 per cent. in a high lime slag, or 16 to 20 per cent. in a slag carrying 12 to 15 per cent. lime, it becomes pasty, and so heavy as not to separate readily from the matte.

A considerable proportion of the zinc oxide escapes slagging and is reduced by glowing carbon in the smelting zone to metallic zinc vapours, which, together with those liberated from zinc sulphide by the agency of metallic

\* Carpenter, *Trans. A.I.M.E.*, vol. xxx., p. 1130.

† Iles, *E. and M. J.*, June 23, 1899, where a description is given of the way in which sulphates of the alkalis have been seen to effloresce from old slag dumps.

‡ *Loc. cit.*, p. 1130.

§ *E. and M. J.*, Jan. 1, 1899, p. 3.

iron, ascend with the furnace gases and condense in the upper part of the furnace. The temperature at which  $\text{ZnO}$  is completely reduced by carbon is said to be  $1,320^{\circ}\text{C}$ ., but reduction begins at a much lower temperature, probably about the boiling point of zinc, which is variously stated at  $920^{\circ}$  to  $1,090^{\circ}\text{C}$ ., certainly there is partial reduction at  $1,100^{\circ}$ . Part of the zinc vapour is re-oxidised by  $\text{CO}_2$  at a temperature slightly below that at which it was reduced, and another part in contact with  $\text{SO}_2$  and carbon may be reconverted into sulphide. The resulting mixtures of metallic zinc, oxide and sulphides of zinc and lead in variable proportions, which condense on the walls (the so-called wall accretions), form exceedingly tough masses most difficult to detach. If not "barred off" in time, they continue to grow, contracting the sectional area of the shaft, and interfering with its regular working so much as to compel the furnace to be blown out.

Such portion of the vaporised zinc as may escape condensation is re-oxidised in the cool upper part of the furnace, and carried into the flue-dust as oxide.

Zinc sulphide is much more objectionable than the oxide, for it passes in much smaller proportion into the slag, and renders it much more pasty. It is important, therefore, to roast blende ores whenever possible before smelting; but even if this be done some zinc always remains as sulphide and a certain proportion will be found in much of the ore treated raw. Zinc sulphide in the charge passes in about equal proportions into the slag and into the matte; where it does harm, not only by lowering the specific gravity of the latter and raising that of the former, but also by rendering both pasty and much less fusible,\* and carrying away into both matte and slag some of the silver which ought to follow the lead. The pastiness of both matte and slag much increases the difficulty of separating them, and gives rise to the production of quantities of foul matte-bearing slag, which has to be re-smelted.

Zinc sulphide is completely decomposed in the theoretical proportions by iron at the melting point of cast iron (say  $1,250^{\circ}\text{C}$ .), and decomposition begins at  $1,167^{\circ}\text{C}$ . Decomposition of  $\text{ZnS}$  by carbon begins at about  $1,200^{\circ}\text{C}$ . When, therefore, the conditions favour powerful reduction, some of the zinc sulphide is acted upon by metallic iron in the hottest part of the furnace with the production of vapour of metallic zinc, which, together with that reduced from the oxide, goes to choke up the cooler part of the furnace.

Zinc in large quantity, therefore, not only increases the difficulty of working the blast-furnace, owing to the pastiness of slag and matte which it causes and the wall accretions which it forms, but actually increases the losses of silver and lead in slags, matte, and flue-dust. Unfortunately, blende is the commonest associate of galena and, therefore, the two minerals should be as far as possible separated before the latter is submitted to the smelting process. To some extent this can be done by dressing operations, but the separation is at best imperfect, entailing much loss of lead and silver in slimes, besides which the blende itself commonly carries a notable proportion of gold and silver, and there always remains a good deal of a "middle product" containing both lead and zinc.

It being, therefore, in a majority of cases impracticable to completely

\* The reason for this no doubt is the insolubility of zinc sulphide in other sulphides, and its comparative infusibility, which causes it to separate out in a crystalline form at temperatures far above the solidification point of the matte, and even of the slag.

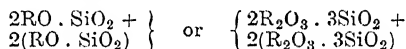
separate the zinc from the lead before roasting, a good deal of attention has been devoted to its separation from the roasted ore before the smelting operation. The various methods for doing this will be found described in Chap. xx. It should be borne in mind, however, that slags containing up to 16 per cent. of zinc, if suitably composed, can be handled without much difficulty,\* especially if most of the zinc is in the form of oxide, provided that the silver contents of the ore-charge amount to less than 1 oz. for each per cent. of lead. With a higher ratio the silver losses become heavy.

#### SLAGS.

**Constitution.**—Slags are mixtures of silicates. The principal slags used in lead smelting are mixtures of monosilicates † in which the oxygen ratio of bases to silica is as 1 : 1, the general formula being—



Bisilicates which are occasionally made in matting practice and in copper smelting, and the general formula of which is  $\text{RO} \cdot \text{SiO}_2$ , are not suitable for lead smelting for reasons which will be seen later on. The sesquisilicate, however, with an oxygen ratio of bases to silica of 2 : 3 is frequently made, and its formula may be written



being regarded as a mixture in equal proportions of monosilicate and bisilicate. Mixtures of mono- and bi-silicates in other proportions are also frequently met with.

Under exceptional circumstances subsilicates, or rather mixtures of mono- and subsilicates, have to be made, in which the oxygen ratio of bases to silica is as  $1 + n : 1$ ;  $n$  being any fraction up to 0.5 or 0.6.

**Physical Properties of Lead Slags.**—The specific gravity of slags has been already referred to. According to Iles, the average specific gravity of one hundred good slags was 3.691, and the general average of the slags produced during six years at the Globe Works was 3.36.‡

As a general rule, 3.25 to 3.65 may be considered as about the practical limits of ordinary lead slags. Seeing that ordinary lead-mattes run from 4.1 to 4.9, the margin for separation is somewhat narrow.

**Crystallisation.**—Most slags crystallise more or less completely in cooling, the crystallisation becoming more perfect as the rate of cooling is slower. When suddenly chilled, as in the operation of sampling (best performed by dipping a clean  $\frac{3}{8}$ -inch iron rod into each slag pot, withdrawing, and plunging quickly into a deep vessel of water), lead slags remain in a vitreous condition, and are quite brittle. When cooled more slowly, on the other hand, they become tough and more or less crystalline. Acid slags are more glassy than the basic varieties, and slags high in alumina and lime remain in a vitreous condition longer than those high in iron and zinc; the latter element in particular has a strong tendency to produce stony slags. It should be borne

\* *v.* Chap. xi., where the practice at the Broken Hill Block 14 Works at Port Adelaide is described in some detail.

† Sometimes called unisilicates or singulo-silicates.

‡ Iles, *Lead Smelting*, p. 131.

in mind that the crystals which develop in the centre of a cooled slag cone are probably often different in composition to the crypto-crystalline matrix surrounding them. Furthermore, it does not appear likely that the mode of crystallisation of lead slags bears any important relation to their fusibility or to slag losses. The author has in his collection two specimens of slag made during the same shift in the same slow-running blast furnace, one specimen being crystallised in perfect cubes and the other in long acicular prisms. Analyses of these two slags are as follows:—

	Cubical.	Prismatic.
SiO <sub>2</sub> , . . . . .	36.95	38.50
FeO, . . . . .	20.89	17.90
MnO, . . . . .	1.68	1.88
CaO, . . . . .	17.85	21.70
MgO, . . . . .	2.64	3.00
Al <sub>2</sub> O <sub>3</sub> , . . . . .	19.74	15.36
S, . . . . .	0.28	0.71
Pb, . . . . .	0.30	trace
	<hr/> 100.33 <hr/>	<hr/> 99.05 <hr/>

The composition and oxygen ratio of these two slags being so similar, it is probable that accidental circumstances rather than the exact percentages of lime, iron, &c., have determined the molecular arrangement and the crystalline form of the silicates. The matter will be again referred to in connection with the subject of aluminous slags.

**Melting Point.**—Schertel determined the melting point of Freiberg (feruginous) lead slag to be 1,073° C. Iles \* conducted some experiments on lead slags by the method of mixtures, using pieces of steel placed in the slag run, and came to the conclusion that the average temperature of slag from a furnace in good work is about 2,000° F. (1,094° C.). The same author quotes Goetz as having found the specific heat of lead slag to be 0.18, and its melting point 2,012° to 2,156° F. (1,060° to 1,180° C.), which confirms the above result. Iles emphasises the fact, well known to all smelters, that the temperature of the slag varies to a great extent independently of the percentage of fuel used, a higher percentage of fuel by no means always giving a hotter slag. The general average temperature of the slags made at the Globe Works from 1895 to 1901 was said to be 1,031° C.†

The formation of the final slag which actually leaves the furnace is a gradual operation, the more fusible combinations of bodies form first and dropping down through the more siliceous and infusible fragments dissolve them away like water trickling through a pile of sugar or of salt.

It might be supposed that a low melting point would always be a most desirable quality, but this is by no means the case. The quantity of fuel required per ton of ore is, indeed, less in proportion as the slag is more fusible, but (since the temperature before the tuyeres can never rise much above the melting point of the slag) both the reducing power of the furnace and the separating power of matte from slag are less in proportion to the lower temperature of the slag. Besides the cost of fluxes, therefore, the necessity of securing sufficient reduction, and a sufficiently clean separation of matte, interfere to some extent with the formation of the most fusible slag possible.

\* *S.M.Q.*, vol. xvii., pp. 20-25.

† Iles, *Lead Smelting*, p. 132.



All slags when cooling pass through a more or less well defined viscous stage before solidifying, though, in the case of a well-proportioned slag of definite composition, the change from the perfectly liquid to the solid condition is comparatively sharp and well marked. Hence, even a good slag may appear thick or viscid, if the furnace is running "cold" from any cause, such as a slight deficiency of fuel or wet charges (and especially wet coke). In such a case it is necessary to charge in more fuel. Generally speaking, slags high in silica or alumina and lime are less perfectly fluid when hot than those high in iron; the former also soften much below the melting point, whereas basic ferruginous slags, which are perfectly fluid above their melting point, set and become stony immediately they cool below this point.

**Formation Temperature.**—Before entering upon a detailed consideration of slag composition, reference must be made to one point which is all important to the practical metallurgist—namely, that the "formation temperature" or softening point of a slag is often and, indeed, generally quite different from the temperature at which it becomes fluid enough to flow freely. A very valuable series of experiments upon the formation temperatures of slags was made by Hofman,\* but a casual reference to the results obtained might be positively misleading to the inexperienced metallurgist without due regard to this circumstance.

As a rule, it may be said that in the case of subsilicates there is no difference between the formation temperature and the true melting point, for such slags pass through no transition stage of softening, and a rise of only a few degrees suffices for their transformation from the condition of a rigid solid to that of a mobile liquid. The divergence, however, between formation temperature and fluidity point becomes more pronounced with increase in the silicate degree; so that, while mixtures of monosilicates somewhat resemble the sub-silicates and approach them in the suddenness with which they attain a highly fluid condition, mixtures of bisilicates, on the other hand, soften gradually, and, like glass, pass through a condition of viscous semi-fusion before attaining sufficient fluidity to flow properly. Frequently, therefore, a slag of this type may require to be heated up to  $100^{\circ}$  or  $150^{\circ}$  above its softening point, or even more, before it becomes fluid enough to flow readily, or to allow of a proper separation from lead and matte.

The difference between the formation temperature and melting point is especially great in the case of the silicates of lime, as proved by the researches of Day and Shepherd,† who found that the bisilicate of lime,  $\text{CaO} \cdot \text{SiO}_2$ , could be formed at  $1,250^{\circ}$  C., whereas its melting point is  $1,512^{\circ}$  C., while the monosilicate,  $2\text{CaO} \cdot \text{SiO}_2$ , could be formed below  $1,700^{\circ}$ , although its melting point is  $2,080^{\circ}$ . Hofman found‡ that the formation temperature of both the bi- and tri-silicates was  $1,250^{\circ}$  C., while the monosilicate was formed at between  $1,250^{\circ}$  and  $1,300^{\circ}$ . In all cases combination begins at a much lower temperature, little above  $1,000^{\circ}$  in fact.

In spite of the fact, therefore, that the formation temperatures and softening points of mixtures of bisilicates are lower, they require actually a higher degree of heat to overcome their viscosity and enable them to flow

\* *Trans. A.I.M.E.*, vol. xxix., p. 683.

† *Journ. Amer. Chem. Soc.*, vol. xxviii., No. 9, p. 1089 (Sept. 1906).

‡ *Bull. A.I.M.E.*, Jan. 1909, pp. 63-65.

properly than do the monosilicate slags. For this reason, coupled with the greater furnace losses both of lead and of silver to which they give rise, bi-silicate slags, although often met with in copper smelting, particularly in reverberatory practice, are never employed in lead smelting.

Sesquisilicates, however, and other mixtures of mono- and bi-silicates with oxygen ratios of 3 : 4, 4 : 5, 5 : 7, &c., are often made. These higher silicates, although they soften at a lower temperature than monosilicates, require a somewhat higher temperature to make them properly fluid. Besides the economy of fluxes, therefore, which results from their use, and which is, of course, the principal object for employing them, their use is desirable whenever a specially high temperature is required in the smelting zone of the furnace, as—e.g., (1) for reducing  $\text{Fe}_2\text{O}_3$  to Fe for use in cleaning the matte of a charge high in sulphur; (2) for smelting highly arsenical ores—the “speiss” produced from which is liable to “freeze” unless kept very hot.

**Formation Temperatures of Slags affected by Nature of Bases.**—The formation temperatures of the various silicate degrees of pure iron slags have been determined with considerable accuracy, and are given below. The corresponding pure lime silicates are so comparatively infusible that their exact melting points have not yet been observed experimentally. They have, however, been calculated approximately from the total heat units contained in a given weight of molten slag, and the most reliable figures seem to be those of Akerman,\* which are given here for comparison with the iron silicates.

	Iron.	Lime.
Subsilicate, $3\text{RO} \cdot \text{SiO}_2$ , . . .	1220°	...
Monosilicate, $2\text{RO} \cdot \text{SiO}_2$ , . . .	1270°	1570°
Sesquisilicate, $4\text{RO} \cdot 3\text{SiO}_2$ , . . .	1120°	...
Bisilicate, $\text{RO} \cdot \text{SiO}_2$ , . . .	1110° +	1512° ‡
Trisilicate, $2\text{RO} \cdot 3\text{SiO}_2$ , . . .	...	1460°

There is an extraordinary difference of opinion as regards the exact melting point of the pure silicates of lime, for Vogt § gives 1,250° as the melting point of wollastonite or lime bisilicate,  $\text{CaO} \cdot \text{SiO}_2$ , which is certainly below the truth. Other authors give various other figures, Le Chatelier, for example, gave 1,700°. While Akerman's and Stein's figures are probably nearer the truth than any others, it seems desirable that careful experiments should be made to determine the exact melting points of all the lime silicates.

The valuable researches of Hofman || have thrown considerable light on the alterations in formation temperature produced by substitution of one base for another. For details, reference should be made to the original paper, but the effect of substituting lime for part of the iron contents of simple iron silicates of different degrees is well shown in Fig. 72. It is seen that simple silicates of iron, from the subsilicate with oxygen ratio of 3 : 2 up to the monosilicate with 1 : 1, have formation temperatures ranging from 1,230° up to 1,280°. Substitution of lime for part of the iron lowers these formation

\* Quoted by Peters, *Principles of Copper Smelting*, p. 298.

+ G. Stein in *Sprecksaal*, 1908, xli., pp. 199-201, gives 1500°-1550°, which, however, is probably incorrect.

‡ Stein (*loc. cit.*) gives 1512°, Akerman gives 1537°.

§ Richards, *Metallurgical Calculations*, p. 113.

|| *Trans. A.I.M.E.*, vol. xxix., p. 683.

temperatures to a minimum of  $1,140^{\circ}$  for the monosilicate of iron and lime with 35 per cent. CaO, above which point further increase in lime causes a

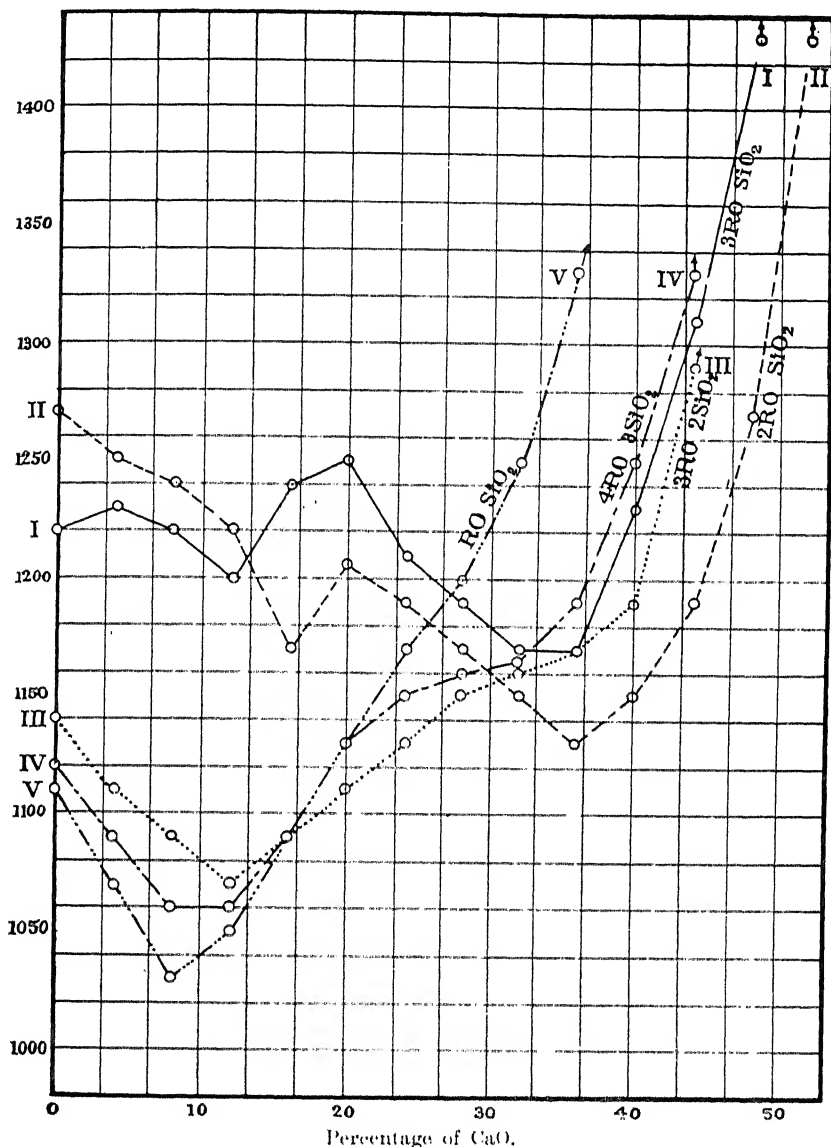


Fig. 72. Formation temperatures of iron-lime silicates.

rapid rise in the formation temperature, so that the monosilicate with 50 per cent. CaO melts at  $1,340^{\circ}$  C.

The more siliceous iron silicates—*i.e.*, those ranging from the 3 : 4 through

the 2 : 3 or sesquisilicate up to the bisilicate with oxygen ratio of 1 : 2, behave quite differently. With iron for their only base the formation temperatures of these slags are nearly alike—viz., from  $1,115^{\circ}$  to  $1,140^{\circ}$ —and substitution of lime for part of the iron at first lowers these formation temperatures by about  $70^{\circ}$  for an addition of about one-tenth of the slag, or, say, one-fourth of the FeO present. In the case of the bisilicate, the minimum formation temperature of all iron-lime silicates—namely,  $1,030^{\circ}$  C.—is reached when the lime amounts to 8 per cent. of the slag, or about one-sixth of the FeO present.

Further additions of lime, however, cause a very rapid rise in the melting point, and the bisilicate in which three-fourths of the FeO has been replaced by CaO is only formed at about  $1,350^{\circ}$  C. Between 24 and 36 per cent. CaO, or when about one-half of the FeO in a slag is replaced by CaO, lime-iron slags of all compositions, from the subsilicate,  $4\text{RO} \cdot \text{SiO}_2$ , up to the bisilicate,  $\text{RO} \cdot \text{SiO}_2$ , have about the same formation temperature—viz., between  $1,160^{\circ}$  C. and  $1,200^{\circ}$  C.—the subsilicate being a little the lowest and the bisilicate the highest. Above this proportion of CaO the melting points of all the silicates are raised with further increase of that base.

Ashley,\* in discussing Hofman's results, adduces theoretical evidence of the existence of a definite chemical compound having the formula  $\text{CaO} \cdot 2\text{FeO} \cdot \text{SiO}_2$ , which he supposes to mix with  $\text{FeO} \cdot \text{SiO}_2$  and with  $\text{CaO} \cdot \text{SiO}_2$  in all proportions, but the evidence for the existence of this body seems to be slender.

When considering all experiments of the kind, it should be borne in mind that the formation temperature of a slag made from an intimate mixture of the various ingredients in suitable proportions, although probably only a little higher than the melting point of the same slag once formed, may be, and often is, very much lower than the temperature at which it becomes sufficiently fluid to flow readily from the furnace and to admit of proper separation of the molten products, the difference being most considerable in the case of the viscous bisilicate and aluminous slags. It should also be borne in mind that owing to the high specific heat of  $\text{SiO}_2$  (0.183) as compared with FeO (0.146) the total heat units required to melt a given quantity of bisilicate slag will frequently be more than would be required to melt the same quantity of a monosilicate slag, although the formation temperature and melting point of the former may be lower. Moreover, neither the actual melting point of a slag nor the total units of heat required to melt it are in practice so important from the point of view of the furnace manager as the density and viscosity of the slag. The density is lowered by increase of either silica or lime, more by the latter than the former. Increase of silica, however, while tending to reduce density, even if the formation temperature be not immediately raised, so increases the viscosity as to interfere with the separation of the molten products; and the fuel consumption has to be increased in order to superheat the slag and bring it to the required degree of fluidity.

**Chemical Composition of Lead Slags.**—We have seen that lead slags are chiefly mixtures of monosilicates or of monosilicates and bisilicates, and that the chief bases employed are iron and lime. These may be replaced to a certain extent only by the corresponding molecular weights of other

\* *Trans. A.I.M.E.*, vol. xxxi., p. 863.

bases. The following table, given originally by Balling,\* shows the proportions of silica required to form monosilicates and bisilicates with different bases. Baryta and zinc oxide have been added, and the figures for sesquisilicates omitted, since they may be regarded as simple mixtures of mono- and bi-silicates in equal proportions. Any required intermediate mixture can be readily obtained from the figures of the mono- and bi-silicates.

TABLE XVI.—RELATIVE EQUIVALENCE OF DIFFERENT  
SLAG FORMING BASES.

One part by weight of Silica requires	Parts by weight of Bases.	One part by weight of Base requires	Parts by weight of Silica.
<i>For Monosilicates—</i>		<i>For Monosilicates—</i>	
Ferrous oxide, . . .	2.40	Ferrous oxide, . . .	0.416
Manganous oxide, . . .	2.36	Manganous oxide, . . .	0.422
Lime, . . . . .	1.86	Lime, . . . . .	0.535
Baryta, . . . . .	5.10	Baryta, . . . . .	0.196
Magnesia, . . . . .	1.33	Magnesia, . . . . .	0.750
Alumina, . . . . .	1.14	Alumina, . . . . .	0.873
Zinc oxide, . . . . .	2.70	Zinc oxide, . . . . .	0.370
<i>For Bisilicates—</i>		<i>For Bisilicates—</i>	
Ferrous oxide . . . . .	1.20	Ferrous oxide, . . . . .	0.883
Manganous oxide, . . . . .	1.18	Manganous oxide, . . . . .	0.845
Lime, . . . . .	0.93	Lime, . . . . .	1.070
Baryta, . . . . .	2.55	Baryta, . . . . .	0.392
Magnesia, . . . . .	0.66	Magnesia, . . . . .	1.500
Alumina, . . . . .	0.57	Alumina, . . . . .	1.747
Zinc oxide, . . . . .	1.35	Zinc oxide, . . . . .	0.740

Table XVII., based primarily upon that given by Hofman, with some alterations and considerable additions, shows the composition of certain typical slags which have been used in practice under different conditions with good results.

All the slags noticed have been in daily use over long periods, and it is local conditions—chiefly the nature of the preponderating supply of ore and fluxes—which have determined the adoption of one or another. Of the whole list, it will be seen that one only, No. 13, is a subsilicate, several are monosilicates, many more are between mono- and sesqui-silicates, and two, Nos. 9 and 21, are bisilicates. The preponderance of siliceous ores and scarcity or high price of fluxes may justify, in exceptional cases, the adoption of a bisilicate slag, but such slags always carry away a considerable amount of lead as silicate, besides being too viscous for good separation from lead and matte, and, moreover, with such slags the rate of smelting is much retarded. Monosilicate slags, especially if high in lime, are ideal as regards fluidity and separation from shots, but they are uneconomical of flux. There is a decided advantage in aiming at a composition approaching that of a sesquisilicate; for although, on the one hand, the fuel consumption is increased, owing to the higher fusing point and the higher specific heat of slag containing an increased proportion of silica, yet, on the other hand,

\* *Compendium der Metallurgischen Chemie*, Bonn, 1882, p. 98.

TABLE XVII.—COMPOSITION OF TYPICAL LEAD SLAGS.

	Authority.	Oxygen ratio Bases : Silica.	SiO <sub>2</sub> .	FeO MnO.	CaO BaO MgO.	Al <sub>2</sub> O <sub>3</sub> .	ZnO.	Total.	Remarks.
1	Eilers, . . . .	1 : 1	28	50	12	...	...	90	} Al <sub>2</sub> O <sub>3</sub> + ZnO under 7 %
2	Eilers, . . . .	1 : 1	30	40	20	...	...	90	
3	Schneider, . . .	1 : 1	33	33	24	...	...	90	
4	Raht, . . . .	1 : 1	35	27	28	...	...	90	
5	Hahn, . . . .	1 : 1.25	34	50	12	...	...	96	} Al <sub>2</sub> O <sub>3</sub> + ZnO under 2 %
6	Hahn, . . . .	1 : 1.40	36	40	20	...	...	96	
7	Murray, . . . .	1 : 1.50	39	32	25	...	...	96	
8	Schneider, . . .	1 : 1.38	34	24	30	10	...	98	
9	Kochler, . . . .	1 : 2	40	26	16	10	4	96	...
10	Page, . . . .	1 : 1.4	33	36	16	...	7	92	} Al <sub>2</sub> O <sub>3</sub> under 5 %
11	Ropp, . . . .	1 : 1.3	30	36	16	...	10	92	
12	Greenway, . . .	1 : 1	25	33	14	3	16	91	
13	Lower Harz, . .	3 : 2	17	35	7	6	20	85	
14	Globe, . . . .	1 : 1.23	31	34	20	6	6	97	...
15	Mapimi, . . . .	1 : 1.06	28	44	15	3	6	96	Pb 2½ %
16	Aguascalientes, .	1 : 1.32	32	35	18	3	7	95	...
17	Monterey No. 1.,	1 : 1.37	32	31	19	7	7	96	Pb 1.0 %
18	San Luis Potosi, .	1 : 1.55	34	28	18	8	8	96	Pb 1.2 %
19	Monterey No. 2.,	1 : 1.5	35	31	23	3	3	95	Pb 1.0 %
20	Dapto, . . . .	1 : 1.3	33	35	17	5	6	96	Pb 1.0 %
21	Freemantle, . .	1 : 2	39	26	15	12	4	96	Pb 1.75 % upwards.
22	Port Pirie, . .	1 : 1.55	25	39	12	6	13	95	...
23	Laurium, . . .	1 : 1.16	25	33	17	12	10	97	Pb 0.8 %

the quantity of slag produced is less, and, therefore, with the same slag assay the losses of lead and silver will be smaller; besides which the higher temperature of the molten slag assists in keeping the crucible in good condition.

In the above Table the oxygen ratio of bases to silica has been calculated on the basis of iron (manganese) and lime (magnesia and baryta) only; the other bases, zinc and alumina, being disregarded in accordance with the general custom of metallurgists, owing to the fact that the behaviour of alumina in slags is neither distinctively acid nor basic. Some metallurgists always calculate zinc oxide also as a base combining with silica and replacing lime, but this is often misleading, particularly if the percentage of zinc is high and that of silica low; for ordinary conditions it is, in the author's opinion, best to consider both zinc and alumina as not combining with silica, but remaining in solution as neutral, although not by any means negligible, components.

Unless the proportion of zinc oxide rises above 10 per cent. the slag should not run below 30 per cent. SiO<sub>2</sub>; since, in order to get it so low, ferruginous flux has usually to be employed, which, besides its cost, raises the specific gravity of the slag, and so increases its liability to carry away, not only matte, but even lead itself. As the proportion of zinc rises, the proportion of silica may be allowed to run down to 25 per cent., and with very barytic ores to as little as 14 or 15 per cent. In ordinary circumstances, however, and with most common mixtures of ores not too rich in zinc, the slags should contain between 30 and 35 per cent. silica.

TABLE XVIII.—FORMULÆ AND COMPOSITION OF TYPICAL MONOSILICATE LEAD SLAGS.

Formulae.	SiO <sub>2</sub> .	FeO.	CaO.	Total.
1. "Quarter" Slag. 3 [2FeO . SiO <sub>2</sub> ], } + 2CaO . SiO <sub>2</sub> , }	30.6	55.2	14.2	100
=	26	47	12	85
Add 5 per cent. SiO <sub>2</sub> ,	31	47	12	90
In practice,	30	48	12	90
2. "Half" Slag. 3 [2FeO . SiO <sub>2</sub> ], } + 2[2CaO . SiO <sub>2</sub> ], }	31.4	45.2	23.4	100
=	27	38	20	85
Add 5 per cent. SiO <sub>2</sub> ,	32	38	20	90
In practice,	32	38	20	90
3. "Three-quarter" Slag. 2FeO . SiO <sub>2</sub> , } + 2CaO . SiO <sub>2</sub> , }	31.9	38.3	29.8	100
=	27	33	25	85
Add 5 per cent. SiO <sub>2</sub> ,	32	33	25	90
In practice,	33	33	24	90
4. 1 : 1 Slag. 3 [2FeO . SiO <sub>2</sub> ], } + 4[2CaO . SiO <sub>2</sub> ], }	32.3	32.3	34.4	100
=	28	28	29	85
Add 5 per cent. SiO <sub>2</sub> ,	33	28	29	90
In practice,	34	28	28	90

TABLE XIX.—FORMULÆ AND COMPOSITION OF TYPICAL LEAD SLAGS.

Number,	5.		6.		7.	
Formulae,	3 [2FeO . SiO <sub>2</sub> ] + 2[CaO . SiO <sub>2</sub> ].		3 [2FeO . SiO <sub>2</sub> ] + 4[CaO . SiO <sub>2</sub> ].		2FeO . SiO <sub>2</sub> + 2[CaO . SiO <sub>2</sub> ].	
Oxygen Ratio,	4 : 5.		5 : 7.		2 : 3.	
	Theory.	Practice.	Theory.	Practice.	Theory.	Practice.
SiO <sub>2</sub> ,	35.5 = 34	34	39.0 = 38	36	41.3 = 39	39
FeO,	51.2 = 49	50	40.2 = 38	40	33.0 = 32	32
CaO,	13.3 = 13	12	20.8 = 20	20	25.7 = 25	25
Total,	100.0 = 96	96	100.0 = 96	96	100.0 = 96	96

**Formulae of Type-slugs.**—The author is of opinion that much may be learned by attempting to reduce to formulæ lead and other slags, and has calculated in the following pages what he holds to be the probable proximate constituents of the slag, the percentage composition of which is given in Table XVII. When the ores are comparatively low in zinc and alumina, so that the total of ZnO, Al<sub>2</sub>O<sub>3</sub>, PbO, S, &c., only makes up about 10 per cent. of

the slag, leaving 90 per cent. for  $\text{SiO}_2$ ,  $\text{FeO}$  ( $\text{MnO}$ ), and  $\text{CaO}$  ( $\text{MgO}$ ), a convenient rule is to calculate  $\text{SiO}_2$ ,  $\text{FeO}$ , and  $\text{CaO}$  to 85 per cent. of the slag, adding an extra 5 per cent.  $\text{SiO}_2$  for the 10 per cent. of bases  $\text{Al}_2\text{O}_3$ ,  $\text{ZnO}$ ,  $\text{PbO}$ , &c. The first four leading monosilicate types calculated thus are given in Table XVIII. (the names "quarter," "half," "three-quarters" simply refer to the ratio of lime to iron, the latter being unity).

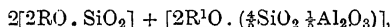
When alumina and zinc are practically absent, so that  $\text{SiO}_2$ ,  $\text{FeO}$ , and  $\text{CaO}$  add up to 96 per cent., and when  $\text{MgO}$  and  $\text{BaO}$  are each under 3 per cent., combinations of monosilicates (iron) with bisilicates (lime) can be adopted advantageously without risk of making the slags pasty. The three type slags in Table XIX. are Nos. 5, 6, and 7 of Table XVII.

**Zinciferous Slags.**—When the proportion of zinc oxide is from 4 to 8 per cent. it may replace lime in any of the four monosilicate slags, 10 parts  $\text{ZnO}$  being equivalent in neutralising power to 7 parts  $\text{CaO}$ . In practice some metallurgists reckon that each 5 parts  $\text{ZnO}$  require the lime to be reduced by rather more than an equal amount, and the iron augmented to correspond. In this way the "three-quarter" slag No. 3 (with  $\text{SiO}_2$  33,  $\text{FeO}$  33,  $\text{CaO}$  24, and  $\text{ZnO}$ , perhaps, 1 or 2 per cent.) becomes No. 10 (Table XVII.) ( $\text{SiO}_2$  33,  $\text{FeO}$  36,  $\text{CaO}$  16,  $\text{ZnO}$  7) when running on zincy ores. It is better, however, to diminish the silica also; thus, slag No. 11 (Table XVII.) is a very good one for zincy ores, its formula being  $3[2\text{RO} \cdot \text{SiO}_2] + \text{CaO} \cdot \text{SiO}_2$ , in which R is  $\frac{4}{3}\text{Fe} \frac{1}{3}\text{Zn}$ . In no case is it necessary to take account, as a slag-forming ingredient, of more than a part of the zinc, say, from one-half in charges comparatively low to one-third of it in charges which are 12 per cent. zinc or over, since the remainder is sent into the matte and the flue-dust.

When the proportion of zinc oxide rises above 10 per cent., all ordinary slags become pasty and tend to form accretions above the lead. With a monosilicate slag, according to Hofman, "16 per cent. is the maximum of zinc oxide with which it is possible to run, and this very soon closes up the crucible," but Greenway has found that over 20 per cent. of zinc oxide can be run without much difficulty, provided the slag be rich in iron and that  $\text{SiO}_2$  and  $\text{CaO}$  are kept within very narrow limits, those for the former being between  $23\frac{1}{2}$  and 26 per cent., and for the latter between 12 and 16 per cent. This is very easy when zinc-lead ores are accompanied by much pyrites which serves as a flux after roasting, but when pyrites is conspicuously absent, as at Broken Hill, it is necessary to use a good deal of barren iron ore.

Slag No. 12 (Table XVII.), produced at the *Broken Hill Block 14 Smelting Works*, leaving out the  $\text{ZnS}$  and  $\text{PbO}$  present, yields on calculation the proportions  $42\text{SiO}_2$ ,  $40\text{FeO}$ ,  $6\text{MnO}$ ,  $20\text{ZnO}$ ,  $25\text{CaO}$ , and  $3\text{Al}_2\text{O}_3$ , corresponding with the formula of a monosilicate in which part of the silica is replaced by alumina, and may be written as follows:—

$20[2\text{FeO} \cdot \text{SiO}_2] + 3[2\text{MnO} \cdot \text{SiO}_2] + 10[2\text{ZnO} \cdot \text{SiO}_2] + 9[2\text{CaO} \cdot \text{SiO}_2] + 3[2\text{CaO} \cdot \text{Al}_2\text{O}_3]$   
or more simply,



in which R is  $\frac{2}{3}\text{Fe} \frac{1}{3}\text{Zn}$ , and  $\text{R}^1$  is  $\frac{2}{3}\text{Ca} \frac{1}{3}\text{Mn}$ .

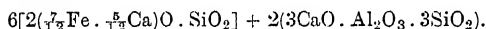
Slag No. 13 (Table XVII.), made at the smelting works near *Goslar* (Harz), averages only 17 per cent.  $\text{SiO}_2$  and, neglecting the  $\text{BaSO}_4$  in solution, corresponds very nearly with the formula:—



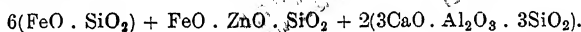


**Aluminous Slags.**—According to Hahn,\* alumina always acts as a base, but almost all metallurgists (including Iles,† Schneider,‡ Henrich,§ and Hofman||) are now of opinion that it plays the part of an acid in slags with a low percentage of silica. As a result of his general study of lead and other slags, the author is of opinion that, when the silica present is insufficient to form monosilicate with the iron and lime, any alumina present acts as a decided acid combined with lime. When the silicate degree ranges from 1 : 1 up to about 1 : 2 alumina is about neutral, and may either be regarded as replacing both bases and silica, or as only lightly combined with lime, zinc, or other bases. Above the bisilicate degree alumina acts as a weak base, and combines definitely with silica, probably, however, as complex silicates together with the other bases, and without affecting the "silicate degree" of the compound, or the oxygen-ratio of bases to silica. It may be noted that both in the pyroxene group of bisilicates and in the amphibole group alumina enters into the combination in a way which cannot be considered as a definite replacement of either acid or base, but rather as a simple addition replacing both; since it is accompanied by no change either in the chemical constitution of the molecule (as represented by its silicate degree and the ratio of the other bases to silica), or in its physical constitution, as illustrated by the crystalline form and optical properties of the mineral. Thus, in the pyroxene group, besides the typical *diopside*,  $\text{CaO} \cdot \text{MgO} \cdot 2\text{SiO}_2$ , we find augites which appear to contain in addition the molecule  $\text{CaO} \cdot \text{Al}_2\text{O}_3 \cdot \text{SiO}_2$ . Similarly, we have amphiboles which may be represented by the formula  $\text{CaO} \cdot 2(\text{MgFe})\text{O} \cdot \text{Al}_2\text{O}_3 \cdot 3\text{SiO}_2$ . In each case alumina enters the combination apparently as a perfectly neutral addition, which in no way disturbs the equilibrium as between acid and base, the compound remaining as a metasilicate; whilst in such a mineral as grossularite, the ultimate formula of which might be approached by an extremely aluminous pyroxene, the alumina clearly exists in the form of an orthosilicate of definite composition.

When the ores are free from zinc or nearly so, and so aluminous as to yield a slag with 8 to 12 per cent.  $\text{Al}_2\text{O}_3$ , and when at the same time iron flux is scarcer and more expensive than limestone, type slag No. 8 (Table XVII.), first made by Schneider, is the proper slag to use. It somewhat resembles No. 4 of Raht, but alumina replaces part of the iron. The formula of this slag worked out from the percentages given is  $12\text{SiO}_2$ ,  $7\text{FeO}$ ,  $11\text{CaO}$ , and  $2\text{Al}_2\text{O}_3$ , which gives :—



The slag formerly made at *Broken Hill* (No. 9 of Table XVII.) was highly aluminous, but it is very doubtful if the alumina played the part of a base as the slag was viscous, and had most of the marked physical characteristics of a bisilicate; for example, the drawing out to an exceedingly fine thread. The formula of this slag works out at  $13\text{SiO}_2$ ,  $7\text{FeO}$  and  $\text{MnO}$ ,  $6\text{CaO}$  and  $\text{MgO}$ ,  $2\text{Al}_2\text{O}_3$ ,  $1\text{ZnO}$ , which may be expressed as—



\* *Mineral Resources of the U.S.*, 1882, p. 328.

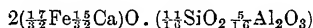
† *Trans. A.I.M.E.*, vol. xi., p. 59.

‡ *Metallurgy of Lead*, p. 143.

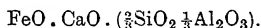
+ *Ibid.*, 1883-4, p. 433.

§ *E. and M. J.*, Dec. 27, 1890.

With still larger proportions of alumina and much less silica the former behaves as an acid, and monosilicate-aluminate slags are formed. Thus, another slag made by Schneider contained  $\text{SiO}_2$  20 per cent.,  $\text{FeO}$  36 per cent.,  $\text{CaO}$  25 per cent., and  $\text{Al}_2\text{O}_3$  15 per cent. = 96. Working out the formula from percentages we have  $11\text{SiO}_2$ ,  $17\text{FeO}$ ,  $15\text{CaO}$ , and  $5\text{Al}_2\text{O}_3$ , corresponding to a simple monosilicate, in which part of the silica is replaced by alumina, thus :—



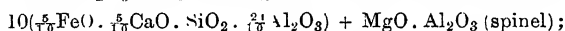
or more simply,



The probable constitution of the remaining type slags which figure in Table XVII. may be represented by the following formulæ :—

14.  $3(2\text{FeO} \cdot \text{SiO}_2) + 2(\text{FeO} \cdot \text{CaO} \cdot \text{SiO}_2) + 5\text{CaO} \cdot \text{SiO}_2 + \text{ZnO} \cdot \text{Al}_2\text{O}_3$
15.  $12(2\text{FeO} \cdot \text{SiO}_2) + 5(2\text{CaO} \cdot \text{SiO}_2) + 2\text{ZnO} \cdot \text{SiO}_2 + \text{ZnO} \cdot \text{Al}_2\text{O}_3$
16.  $8(2\text{FeO} \cdot \text{SiO}_2) + 10(\text{CaO} \cdot \text{SiO}_2) + \text{ZnO} \cdot \text{Al}_2\text{O}_3 + 2\text{ZnO}$
17.  $6(2\text{FeO} \cdot \text{SiO}_2) + 2\text{CaO} \cdot \text{SiO}_2 + 8(\text{CaO} \cdot \text{SiO}_2) + 2(\text{ZnO} \cdot \text{Al}_2\text{O}_3)$
18.  $5(2\text{FeO} \cdot \text{SiO}_2) + 2\text{ZnO} \cdot \text{SiO}_2 + 6(\text{CaO} \cdot \text{SiO}_2) + 2(\text{CaO} \cdot \text{Al}_2\text{O}_3 \cdot \text{SiO}_2)$
19.  $2\text{FeO} \cdot \text{SiO}_2 + 2(\text{CaO} \cdot \text{SiO}_2) + \frac{1}{2}(\text{ZnO} \cdot \text{Al}_2\text{O}_3)$
20.  $10(2\text{FeO} \cdot \text{SiO}_2) + 2(2\text{CaO} \cdot \text{SiO}_2) + 10(\text{CaO} \cdot \text{SiO}_2) + 2(\text{ZnO} \cdot \text{Al}_2\text{O}_3) + \text{ZnO}$
21.  $16(\text{FeO} \cdot \text{SiO}_2) + 8(\text{CaO} \cdot \text{SiO}_2) + 4(\text{CaO} \cdot \text{Al}_2\text{O}_3 \cdot \text{SiO}_2) + \text{ZnO} \cdot \text{Al}_2\text{O}_3 + \text{ZnO}$
22.  $4(2\text{FeO} \cdot \text{SiO}_2) + \text{FeO} \cdot \text{ZnO} \cdot \text{SiO}_2 + 4(\text{CaO} \cdot \text{SiO}_2) + \text{ZnO} \cdot \text{Al}_2\text{O}_3 + \text{ZnO}$
23.  $4(\text{FeO} \cdot \text{CaO} \cdot \text{SiO}_2) + 3\text{FeO} \cdot \text{SiO}_2 + 2(\text{ZnO} \cdot \text{Al}_2\text{O}_3)$

Although the compound,  $\text{ZnO} \cdot \text{Al}_2\text{O}_3$ , represented in many of the above formulæ is found in nature as the mineral *Gahnite*, it is not intended to suggest that that mineral is necessarily present in any of the above slags, but rather to indicate that, as a rule, the silica, iron and lime combine together in their own fashion, and that the zinc and alumina are left in more or less similar condition—namely, as comparatively inert bodies in igneous solution, either free or but lightly combined. Hutchings points out\* that, although zinc oxide is often found in slags after cooling in the form of zinc-iron or zinc-iron-alumina-spinel, it is also quite often found in the form of crystallised *willemite* ( $2\text{ZnO} \cdot \text{SiO}_2$ ), which is here figured as existing in slags Nos. 12, 15, and 18. That compounds of the spinel group may, however, actually exist in many slags is indicated by the fact mentioned by Muirhead,† who found that certain highly aluminous blast furnace slags left upon decomposition with acid a very refractory residue, amounting to from 5 to  $17\frac{1}{2}$  per cent. of the weight of the slag, and that this residue upon analysis proved to be spinel ( $\text{MgO} \cdot \text{Al}_2\text{O}_3$ ) with one-third of the  $\text{MgO}$  replaced by  $\text{FeO}$ . All the minerals of the spinel group crystallise in the cubical system, and it may be the presence of members of this group of aluminates which determines the crystallisation of certain slags in that system. For instance, referring to the two aluminous slags described on p. 150, upon calculating out their formulæ, they are found to be very similar, and it seems probable that the difference in the crystalline form follows a real difference in the molecular arrangement. Thus the cubical crystals, the composition of which is  $10\text{SiO}_2$ ,  $5\text{FeO}$ ,  $5\text{CaO}$ ,  $1\text{MgO}$ , and  $3\text{Al}_2\text{O}_3$ , may perhaps be represented by the formula—

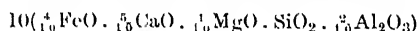


whereas the prismatic crystals, the composition of which is approximately

\* *E. and M. J.*, Dec. 24, 1903.

† *E. and M. J.*, Jan. 18, 1908, p. 164.

$10\text{SiO}_2$ ,  $4\text{FeO}$ ,  $5\text{CaO}$ ,  $1\text{MgO}$ , and  $2\text{Al}_2\text{O}_3$ , may perhaps be written



corresponding with the aluminous lime-iron amphiboles.

**Replacement of FeO and CaO by other Bases.**—A series of experiments was made by Hofman\* on the replacement of FeO and CaO in a typical lead slag by other bases, the slag chosen being that having the percentage composition  $\text{SiO}_2$  32.1, FeO 35.9, CaO 32 = 100, the molecular composition of which is  $10\frac{1}{2}\text{SiO}_2$ , 10 FeO,  $11\frac{1}{2}\text{CaO}$ , and its formation temperature  $1,150^\circ\text{C}$ . The experiments were directed to determining the changes in formation temperature that take place as one constituent is gradually replaced by another, and the results may be summarised as follows :—

*Manganese* substituted for iron increases the fluidity of slags, while slightly diminishing their fusibility, for which decrease, however, the gain in fluidity more than compensates.

*Alumina* may replace ferrous oxide up to the proportion of one-half without perceptibly affecting either the melting point or the viscosity, but above that proportion the melting point rises rapidly. It may also replace CaO up to three-fourths of the total without raising the melting point, but there is a marked increase in viscosity. Up to seven-sixteenths of both bases (*i.e.*, up to a slag having the composition  $\text{SiO}_2$  32, FeO 22, CaO 20,  $\text{Al}_2\text{O}_3$  14), alumina may be substituted for both iron and lime without raising the melting point; the viscosity is, however, always somewhat increased, and above the proportion mentioned both melting point and viscosity increase rapidly.

*Magnesia* may replace lime in proportions up to five-eighths of the CaO present, but apart from its greater saturating power not to advantage, as the melting point is always somewhat raised and the viscosity markedly increased.

*Zinc oxide* may replace lime up to even three-fourths of the CaO present; the melting point is sometimes lowered, though in most proportions raised, and the fluidity invariably diminishes almost *pari passu* with the increase in zinc.

*Baryta* may replace lime in all proportions, and always lowers the melting point without reducing the fluidity, but since in large amounts it increases the specific gravity, it is not desirable as tending to interfere with the separation of the molten products. When, however, considerable amounts of magnesia or zinc oxide are present, addition of baryta notably lowers the melting point without increasing the viscosity produced by the former oxides; and whenever available, therefore, its use in such cases is indicated as desirable.

Hofman's experiments further show that in what he calls his basal slag from one-fourth to three-eighths of the *silica* present may be replaced by alumina with a rise of only  $40^\circ$  in the formation temperature of the slag ( $1,150^\circ$  to  $1,190^\circ$ ), although the viscosity is markedly increased. Above this proportion of alumina both melting point and viscosity increase suddenly.

**Ratio of Iron-bases to Lime-bases.**—In copper smelting and matte smelting the proportion of iron in the slag can be brought quite low, but in lead smelting there would seem to be some minimum ratio of iron-bases to lime-bases which it is not possible to overstep and yet do good work. Thus, some years ago, what is known as the Schneider type of slag,  $\text{SiO}_2$  35, FeO 27,

\* *Trans. A.I.M.E.*, vol. xxix., pp. 682-721 (1899).

CaO 28, was somewhat widely used, particularly where iron fluxing ores were scarce, because it is economical in such cases, and gives low assays for both lead and precious metals. A marked peculiarity of this slag, however, is that of becoming thick while still a good deal above its melting point, and of setting hard quite suddenly, often clogging in the spouts; it is, therefore, exceptionally liable to lead to trouble with the tapping, even if the amount of zinc present is low, but especially if the ZnO rises above 2 or 3 per cent. For this reason, in spite of the low slag assays which can be secured by adopting this composition of slag, few metallurgists like to use it if they can secure more iron. As to the reason for the quick setting of this slag, and for its becoming "stringy," and clogging in the spout, as it often does, especially when the silica accidentally falls a per cent. or two below the theoretical point, the following results of some experiments recently made at an important smelting plant, and covering a period of several weeks, with about six furnaces running, may be of interest. With a view to economy in the use of expensive iron flux, after a little experimenting on one furnace only, all the furnaces were put on a high lime slag for a short time, but in view of the trouble experienced with the slag breasts the proportion of lime was gradually reduced after about a fortnight's run on slag of each of the compositions given below:—

Composition of Slag.	1st Period.	2nd Period.	3rd Period.	Normal Slag.
Silica, . . . . .	35·21	36·31	36·46	35
Iron oxide, . . . . .	26·58	28·15	28·96	31
Lime, . . . . .	26·17	24·08	22·91	23
Total, . . . . .	87·96	88·54	88·33	89
Oxygen ratio—				
Silica : bases, . . . . .	1 : 1·48	1 : 1·50	1 : 1·50	1 : 1·40
Molecular ratio—				
FeO : CaO, . . . . .	1 : 1·25	1 : 1·10	1 : 1·02	1 : 0·945

During the first period, while reduction of lead was good, some of the furnaces were always troubled with hard breasts; during the second period, while this condition was less accentuated, it still occurred with sufficient frequency to be troublesome with the class of furnace labour available; and only when the iron was increased and the silica a trifle reduced were the furnace conditions quite satisfactory. It would really seem as if the ratio of FeO to CaO is of considerable importance, as if, in fact, in lead smelting any increase in the latter above the proportion of 1 to 1 is liable to give trouble and to bring about irregularities in working. It should be mentioned, however, that at the works where these experiments were conducted the furnaces are all low, and it is probable that with higher furnaces slags containing a somewhat higher percentage of bisilicate of lime could be run with equal facility. There must, however, be a minimum percentage of iron for good work in lead smelting, and it is probable that this limit is reached at somewhere about 24 per cent.

## CHAPTER VIII.

### BLAST-FURNACE LEAD SMELTING—PLANT.

**General Arrangement.** The general arrangement of a smelting plant, though perhaps more a question of engineering than of metallurgy pure and simple, is very important; for modern metallurgy is becoming more and more a question of engineering and less of empirical rule of thumb. A blast-furnace plant always requires three levels—viz., those of the ore-delivery floor, feed floor, and slag dump. There should be at least 8 feet, and preferably 12 feet, between the ore delivery floor and the feed floor, 20 to 30 feet, according to the height of furnace adopted, between the feed floor and the slag dump floor, with as much below the last as possible. Where natural fall is available, the ground may be laid out in terraces, as shown in Fig. 73, which shows a general arrangement of furnace, ore-bins, dust-chambers, &c., on a so-called terrace site. It is but rarely that the conformation of the

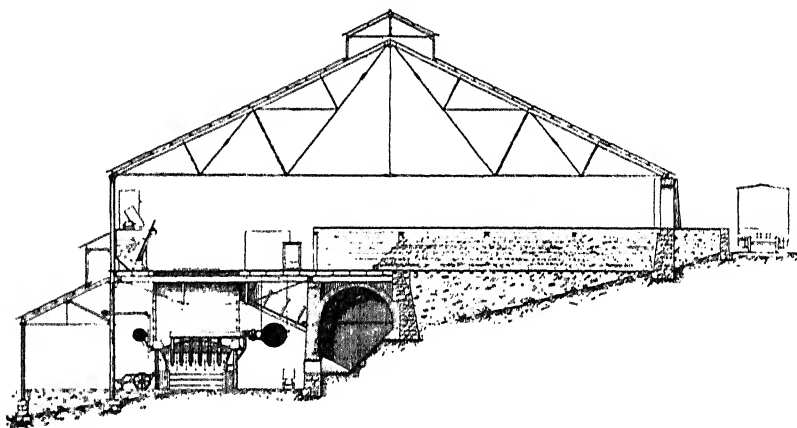


Fig. 73. —Arrangement of Smelting Furnace and Ore-beds on Terraced Site.

ground will suit for laying out any considerable proportion of the ore-bins as shown in Fig. 73, and in default of natural fall elevators or hoists may be used for the feed floor, while the height for ore-bins is got by elevated tracks over them.

The remaining details of arrangement will depend upon the site available, the railroad communications, the composition of the ores to be treated, and the regularity, or otherwise, of their supply; they vary so much that only practical experience can give the student some insight into the question.\*

\* For further information the student may be referred to Locke, *Smelting Plants*, Cincinnati, 1883; and to Hoffman, *Metallurgy of Lead*, 1906, pp. 203 to 222. Detailed descriptions of existing smelting plants are also given in papers by the author and J. W. Malcolmson in *Proc. Inst. Civ. Eng.*, vol. cxiv., part ii.

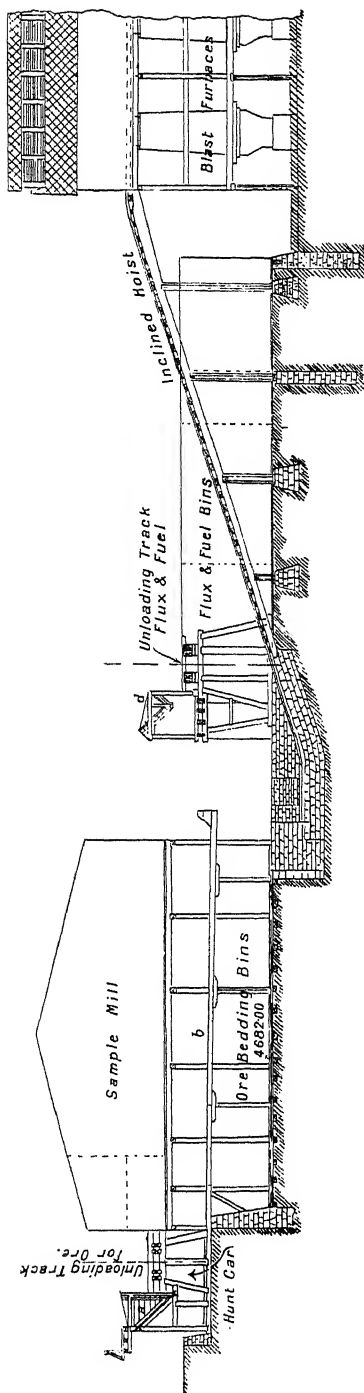


Fig. 74.—General Arrangement Blast-furnace Plant, Pueblo S. and R. Co. (Pueblo, Colo.).

Where the available fall is still less in amount, it must be all utilised for dump space. The ore-bins will then be built above the ground level, covered by elevated tracks reached by long inclines, and the charge will be all elevated to the feed floor.

In the case of a large custom plant situated generally at some important junction of two or more railway systems, the all-important consideration is railway facilities; and, in order to get in convenient sidings at reasonable cost, it is worth while to sacrifice almost all other considerations in regard to site, so that it happens in practice that nearly all such custom plants are built upon more or less flat ground, and the charges are invariably raised to the feed floor. Fig. 74 shows some details of the general arrangement of a plant built upon a flat site—namely, the Pueblo plant of the American Smelting and Refining Co. at *Pueblo*.

**General Consideration of Furnace Construction.**—Lead-smelting blast furnaces were formerly built of brick or stone throughout, rectangular in section, blown by tuyeres in the back wall of the furnace only, and provided with an interior crucible or sump which was tapped at intervals into a “tap-pit” in front of the furnace. They were commonly made wider at the tuyere level than above, and may be seen figured in any of the older works on the metallurgy of lead.\* The great disadvantages of all such furnaces were—(1) *Low temperature in the smelting zone*, causing incomplete reduction of lead and separation from slag and matte. (2) *Comparatively high velocity of the effluent gases*, which caused a high temperature at the mouth of the furnace, an excessive

\* The simplest of all such furnaces actually in use within recent years in comparatively remote parts of Mexico was described and figured by the author in *Trans. I.M.M.*, vol. xii., p. 410.

consumption of fuel, and a very high loss of lead by volatilisation. (3) *Small capacity*, with its inevitable concomitants, high cost of treatment, and great irregularity in working. (4) *Inaccessibility* of the interior and *want of control* over the furnace working except by blowing out.

Such furnaces are now extinct, save perhaps in very remote regions, having been replaced everywhere by higher furnaces, circular or oblong in section, having their walls protected by exterior iron casings, and provided with tuyeres arranged symmetrically round the exterior instead of only at the back. The area at the tuyeres is kept constant, either by the use of water tuyeres (Harz) or water blocks (Przibram), or, more commonly, by water-jacketing the whole smelting zone of the furnace. The two former systems are still in use to a limited extent in parts of Europe, but the latter have so many advantages that their use is to be recommended in every case where sufficient supply of water is obtainable at a reasonable cost. Where water is very scarce and refractory material easily obtained, there is much to be said in favour of the water-block and water-tuyere system, especially when the supply of ore is irregular.

The tuyere area in modern furnaces is almost invariably considerably less than the throat area, which is advantageous in two ways :—(a) because, by keeping the tuyere area small, the temperature of the smelting zone is high, a condition conducive to good reduction of lead and production of clean slags ; and (b) because the widening towards the feed floor causes a diminished velocity in the ascending current of furnace gases, which promotes the absorption by the cold charge of the heat they contain, and greatly reduces the amount of lead carried off as flue dust and fume. As a rule, with circular furnaces, the diameter increases gradually from the tuyeres to the throat, while rectangular furnaces are built with a bosh, which is preferable on account of the clearer definition of the smelting zone thereby secured. In either case the area at the throat should be from two to two and a-half times that at the tuyeres, in order to ensure that the velocity of the gases shall be sufficiently slackened.

Lead furnaces may be charged either from the feed floor level, the fumes being removed by a downtake flue below the floor, or by means of charging-doors in a chimney or hood of brick or sheet iron forming a continuation of the stack, in which case the gases and flue dust are taken off by an over-head downtake. Both arrangements are employed for oblong as well as for circular furnaces, but the former is more common.

The height of modern furnaces varies chiefly with the nature of the ores to be smelted. Ores rich in lead and producing a ferruginous slag are comparatively easy to smelt at a low temperature, which can be obtained with a low pressure of blast, and, consequently, a moderate height of 10 to 14 feet above tuyeres is sufficient. Ores poor in lead and rich in silica, to which a large quantity of limestone has to be added in order to form a calcareous slag, require, on the other hand, a high temperature and long continued heat in the smelting zone, only obtained by the use of a high-pressure blast ; consequently, a greater height of from 16 to 20 feet is required in order to prevent heat from being wasted at the surface of the charge, and also by considerations of the maximum tonnage capacity attainable without increasing the working losses.

The number of tuyeres, formerly from one to three, has been increased both

in circular and in oblong furnaces to from eight to fourteen, according to the size of the furnace. The smallest number now employed in practice is five (Clausthal and some parts of Spain), but whatever the number, they are evenly distributed round the furnace. The diameter of tuyeres varies between  $2\frac{1}{2}$  inches and 4 inches, but 3 inches to  $3\frac{1}{2}$  inches are the sizes most commonly employed. The distance across the furnace between the tuyeres varies from 36 up to about 60 inches, the latter figure, however, being rarely met with. Increasing the size of circular furnaces, of course, results in greater capacity, but hardly in proportion to the increased area, while the pressure of blast has to be augmented in order to ensure penetration to the centre of the charge. The augmented blast-pressure requires, in order to preserve the reducing action, simultaneous increase in the height of the furnace; and is accompanied by the disadvantages of increased loss by volatilisation, and of a creeping upward of the smelting zone which reduces the temperature at the tuyeres. With circular furnaces of moderate height, moreover, the fuel has to be thrown towards the centre in order to allow the blast to penetrate and prevent the formation of a chimney of unmelted charge, while with oblong furnaces fuel, ore, and fluxes may be spread in even layers, which gives better reduction. The capacity of an oblong furnace may be doubled by simply doubling its length, without interfering with its breadth or altering any other condition; whereas doubling the sectional area of a circular furnace does not result in doubling its capacity, unless the height and the blast pressure are increased proportionately at the same time.

For basic ores rich in lead and poor in silver producing fusible ferruginous slags, where the high volatilisation loss is comparatively unimportant, large circular furnaces may be employed with advantage (*e.g.*, Spain); but with ores poor in lead and rich in silver, and especially in cases where the slags produced are high in silica and lime, the oblong form is far preferable (United States, Mexico, Australia). The blast pressure employed with circular furnaces varies according to the height of the furnace from 10 to 20 inches of water (6 to 12 ozs. per square inch) with a diameter of 36 to 42 inches. With larger circular furnaces of from 48 to 60 inches diameter the blast pressure must be increased to 1 lb. or even  $1\frac{1}{2}$  lbs. (28 to 42 inches of water) if the ores are at all infusible. The tonnage capacity of a furnace of given area is much increased by raising the blast pressure, but it is necessary to raise simultaneously the height of the charge column, in order to keep the top cool and to prevent losses by dusting and volatilisation; with modern high furnaces blast pressures ranging up to from 40 to 45 ozs. are commonly employed.

Lead furnaces may be provided with an interior "sump," which is tapped at intervals into an exterior "tapping box," or they may have an interior "well" or "crucible," which is always kept full of lead, and which communicates by means of a channel called the "syphon tap" with an iron pot, from which the lead can be ladled almost continuously. The former is the old arrangement still used in a few German lead furnaces, but now almost everywhere obsolete; the latter arrangement being in almost universal use. One great disadvantage of the sump furnace is that the operation of tapping interferes with the regular working of the furnace, a large mass of unsmelted material being suddenly lowered through the smelting zone, and the sudden contact of solid material with the bottom of the hearth, when the bath of lead is removed by tapping, is liable to give rise to the formation of "sows"



and "bottom crusts," especially as during the tapping operation the blast has to be turned off for a while. It is, too, a disadvantage that the charge suddenly dropped down in front of the tuyeres by the draining off of the bath of molten products by which it is supported, and, coming as it does, from higher up in the smelting zone, takes some minutes to reach the condition of that which was in front of the tuyeres before tapping took place, while the coke which is suddenly dropped into the bath of slag gets its pores filled with molten material, and is not so readily burned when the bath again rises to its normal level. Tapping from a sump, therefore, always interferes to some extent with the regular working of the furnace. Crusts will occasionally form on top of the lead in a crucible furnace, but they are much more readily removed (by taking down a jacket) than is the case with crusts formed in a sump furnace.

The syphon tap for continuous discharge of lead from the crucible is an improvement of a very old form of exterior fore-hearth described and figured by Karsten; it was invented by a Clausthal metallurgist called Arents, and was first tried in Nevada, from whence it spread to every lead smelting district in the world. It is shown in Fig. 83, which represents an American circular water-jacket furnace; it consists simply of a channel built in the fire-brick wall of the crucible and connected at the top with a basin-shaped depression in the same, which is sometimes provided with a cast-iron casing and is called the "lead well." Owing to the pressure on the lead in the crucible caused by the weight of slag and charge floating on it, the level of the lead in the well is always some inches above that of the inside of the crucible, and with charges poor in lead there is some tendency to chill in the tube, which is guarded against by inserting in it from time to time red-hot iron bars. Besides the greater regularity in furnace working and the avoidance of "sows," the use of the "automatic tap" results in a smaller production of dross. It is, however, not to be recommended in the case of very coppery charges (as, for example, in concentrating lead matte containing upwards of 12 per cent. Cu), on account of the great tendency to form a "bottom crust" consisting of an alloy of copper and lead with a high melting point, which gradually grows up from the bottom till it fills the crucible. In these cases, therefore, it is better to abandon the automatic tap and work the furnace as a sump furnace, tapping from the bottom. Sump furnaces are also preferable in cases where ores have to be smelted which are exceptionally rich in zinc and in barium sulphate, like those of Goslar, which could not well be smelted in the ordinary way with an automatic tap, on account of the quantity of crusts which would form above the lead. By tapping from the bottom the hearth can be cleaned up at each tapping and the accretions removed.

The blast used in lead furnaces is almost always produced by a so-called "positive blower" \* with rolling impellers of the Root type. The Baker type of blower was formerly used a good deal in American plants, but the Root type is decidedly better, being not only more efficient, but built more solidly, and because of the much greater margin of speed permissible in running them; taking a Root and a Baker of approximately the same capacity at ordinary speeds and low pressure, the former can, in case of emergency, be worked up to more than double its proper speed without injury and without

\* In contradistinction to the induced current produced by a high-speed fan.

much loss of efficiency, whereas the efficiency of the Baker at high speeds and with pressures exceeding 1 lb. per square inch is very low.

Hot blast as yet finds no use in lead blast furnaces, for the temperatures required are only such as can be economically produced by burning fuel before the tuyeres with cold air. It is generally supposed that the use of hot blast would so increase the reducing effect of the furnace as to give rise to the formation of iron "sows" in the crucible, besides enormously increasing the loss of lead by volatilisation.

It may be convenient to consider first the circular furnaces, which are still commonly employed in Europe, and then to describe in greater detail the large oblong furnaces which are preferred in the United States, Mexico, and Australia.

The comparatively speaking primitive furnaces of small capacity are here described at greater length than their present importance would seem to warrant, because there are still many remote districts where, in consequence of difficulties of transport and comparative scarcity or irregularity of the ore supply, only such furnaces can be employed. Individual mines in out-of-the-way places, producing only small amounts of ore suitable for smelting, must continue to employ just such small furnaces; and a study of the various types will enable any young metallurgist to improvise for himself out of simple materials a furnace suitable to the conditions.

#### CIRCULAR FURNACES.

The following types may be figured and described :—

1. The Castilian or Spanish blast furnace with water-tuyeres.
2. The Lower Harz furnace with simple water-tuyeres.
3. The Clausthal furnace with water-tuyeres and air-cooled boxes.
4. The Przibram furnace with water-tuyeres and water-blocks.
5. The Freiberg Pilzofen with water-jacket in sections, the prototype of all water-jacketed lead furnaces.
6. The American round furnace with water-jacket in one piece and overhead downtake.

Of these the first three are sump furnaces, while the latter two are provided with automatic discharge.

1. The Spanish blast furnace, "*Castellano*" or "*pavo*," is in some respects the most primitive of all types of furnace at present at work in civilised countries. It is built either of brick, or more commonly of sandstone, grit, or other refractory stone throughout, the zone of fusion being in any case built of blocks of refractory stone cut to the proper shape. An ordinary thickness for this masonry on first blowing in is 2 feet at bottom and 16 inches towards the upper part of the shaft; but although the total length of a campaign may be from six to ten weeks, fire generally shows through the masonry of the fusion zone within a week after blowing in, and thereafter work is carried on by means of patching with clay and by replacing badly burnt-out stones here and there with new ones cut to size.

Figs. 75 and 76 show in section and in plan respectively one of the simplest types of Spanish furnace with five tuyeres; more or less similar furnaces with five and six tuyeres may be seen at work at *Cordoba*, *Linares*, and many other places.

At *Peñarroya* and other establishments much larger rectangular “pavos” are employed having seven and nine tuyeres, built of brick throughout, and run with a much higher charge column.

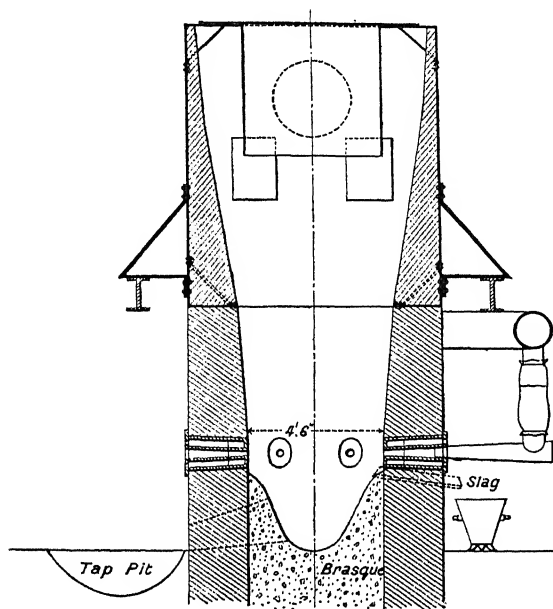


Fig. 75.

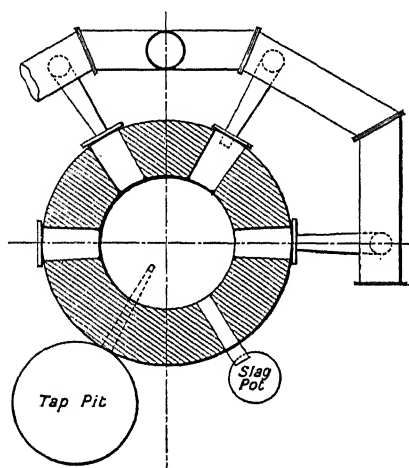


Fig. 76.

The capacity of these various forms of the Spanish furnace varies according to the size from 25 or 30 tons for the circular “castellanos” up to

100 tons of charge per day for the large rectangular "pavos." The charges treated are generally rich, say from 25 to 40 per cent. lead, and at a large number of works where rich galena ores are treated in reverberatories or in hearths blast furnaces of this general type are in use for reducing slags and other bye-products, together with second-class ores.

2. **The Lower Harz Furnace.**—Except the Spanish, this is the most primitive type of furnace now at work. Its construction will be seen from the vertical section, Fig. 77, and the horizontal section through tuyeres,

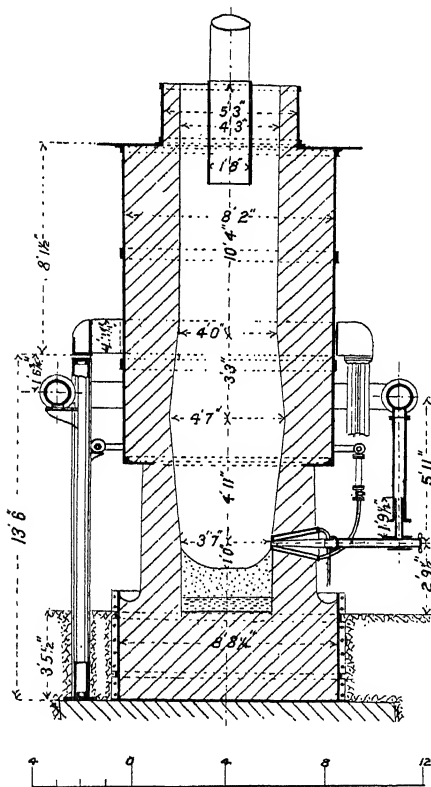


Fig. 77.—Section.

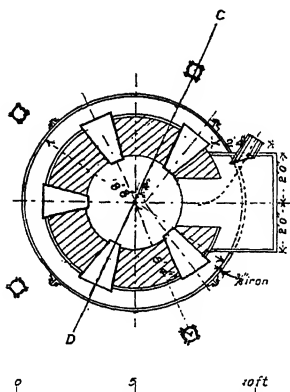


Fig. 78.—Plan at tuyere level.

Fig. 77 and 78.—Lower Harz Furnace.

Fig. 78. The sump is formed of "steep" (brasque) rammed inside the brick foundation, which is enclosed in iron plates to prevent leakage of lead. The upper portion of the furnace lining, above the smelting zone, wears very little, and is entirely supported on iron plates, forming a continuation of the outer casing, the whole of which rests on the four columns shown in the plan. The lining in the smelting zone, which wears out very fast, is thus easily repaired without interfering with the upper portion of the furnace. The height from tuyeres to throat is 18 feet 6 inches, and the tuyere level is 1 foot above

the bottom of the hearth. The shaft is contracted immediately above the bosh in order to prevent, as far as possible, the formation of zinc accretions, by accelerating the speed of the ascending current of furnace gases and zinc fumes. The gases are drawn off by a tube of comparatively small diameter, suspended in the centre of the shaft; the advantage of which is that the current of gases is thus attracted towards the centre and away from the sides, where it is liable to deposit zinc accretions.

The water-tuyeres shown in the figures are inclined at an angle of  $3^\circ$ , and are 2 inches in diameter; they are five in number, and disposed symmetrically round the perimeter of the furnace, as shown in the plan. The blast pressure is about  $1\frac{1}{4}$  to  $1\frac{3}{8}$  inches of mercury, and each furnace treats  $8\frac{1}{2}$  to 10 tons of ore per day (*v. Chap. xi.*).

3. **The Clausthal furnace** is somewhat similar to the above in construction, but different in form. Instead of a bosh, it gradually increases in diameter from tuyeres to throat (Fig. 79). The whole furnace is built upon a stone foundation block, F, which supports a cast-iron foundation plate; the crucible is built of firebrick, enclosed by cast-iron plates, and encloses a succession of hearth layers, the uppermost of which is of brasque. The lining of the shaft is firebrick surrounded by a casing, *b*, of boiler-plate, to which are rivetted angle irons, *a*, by means of which the shaft is supported on hollow columns resting in cast-iron sockets, as shown. The furnace gases are drawn off through the tube *g*; the furnace has four water-tuyeres, *t*, each  $2\frac{1}{2}$  inches diameter, supplied with air at 6 to 8 ozs. pressure. These water-tuyeres pass through cast-iron cooling boxes, O, which are cooled, partly by a current of air, and partly by the circulation of water in the water-tuyeres. This seems a very imperfect arrangement, and the use of complete water-jackets is certainly preferable; the practice in these furnaces of the "precipitation" process, elsewhere obsolete, is also peculiar, and will be described in Chap. xi.

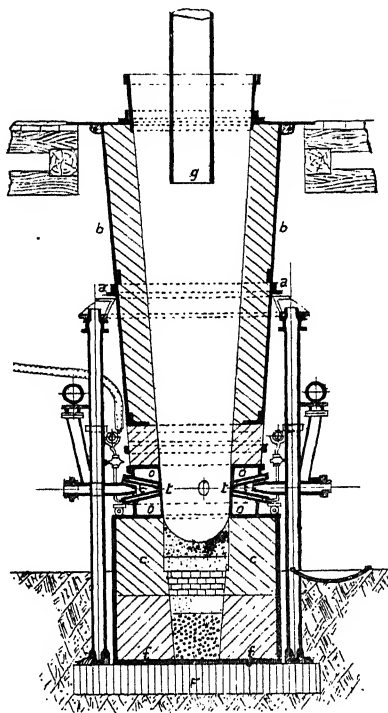
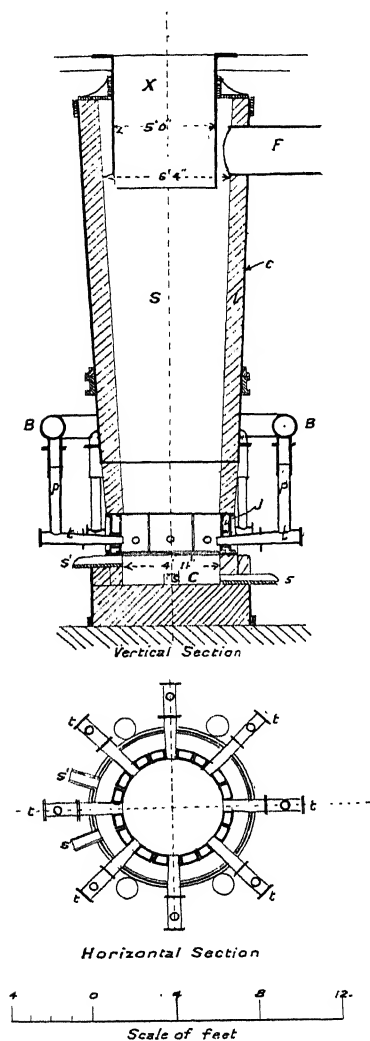


Fig. 79.—Clausthal Furnace.

4. **The Przibram Furnace.**—This furnace, shown in section in Fig. 80, is built somewhat after the pattern of an iron blast furnace in miniature. The walls are thick and double, with an expansion space between the inner firebrick lining and the outer wall of common brick; there is no exterior iron casing. The upper part of the furnace rests upon a framework of flanged cast-iron plates supported by six cast-iron columns, *a*; the smelting zone of firebrick is protected by three rows of cast-iron "water-blocks." The lower



The water-jacket of this furnace is of wrought iron, and is built up in segments, each segment having a tuyere aperture in the centre. The construction of the furnace is shown in Figs. 81 and 82. *S* is the shaft, *c* the casing of boiler-plate surrounding the brick lining, *l*, which rests on four cast-iron



Figs. 81 and 82.—Pilz Furnace.

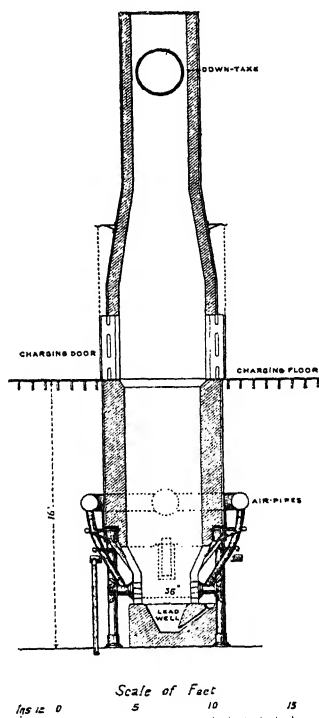


Fig. 83.—American Circular Water-jacket Furnace.

columns. *J* are sections of the jackets, each 8 inches wide and 20 inches high, with an iron tube through which are inserted the tuyere nozzles, *t*. Cold water enters through a small pipe opening into the bottom of each section, while the hot water is discharged through another pipe at the top.

row contains eight water-tuyeres, which are simply cast-iron boxes with a conical bushing cast through the centre into which fit the sheet-iron tuyere nozzles. The water-block over the slag spout, *s*, is of a peculiar shape. The two upper rows are composed of larger water-blocks, *b*, of cast iron (Fig. 80B) alternating with brick, as shown in Figs. 80 and 80A. The crucible of

the furnace is composed of layers of rammed "brasque" on a layer of brick, under which again are other layers of sand and loam, the whole being supported on a massive cast-iron foundation plate, *f* (Fig. 80), to avoid leakage of lead. The discharge of lead from the crucible takes place automatically by means of Arents' syphon tap (not shown in the figure) as in American furnaces. Charging is performed by means of a cast-iron cup, *c*, with sheet-iron curtain hanging from it, the furnace gases being taken away by the flue, *F*, below the charging floor.

One of these furnaces with 2½-inch tuyeres supplied with air at a pressure of 1·6 inches of mercury smelts in twenty-four hours 31 tons of charge (=22 tons of ore), with a coke consumption of 16·4 per cent. and about  $\frac{3}{4}$  per cent. of charcoal ( $1\frac{1}{2}$  bushels per ton). Further details as to the working of the furnace are to be found in Chap. xi.

5. **The Pilz furnace**, invented at Freiberg by Bergrath Pilz, in 1863, is undoubtedly the first complete water-jacketed furnace ever used for smelting lead ores, and the prototype of the American water jackets. It is equally certain that at Freiberg, under Government control, the furnace has undergone but

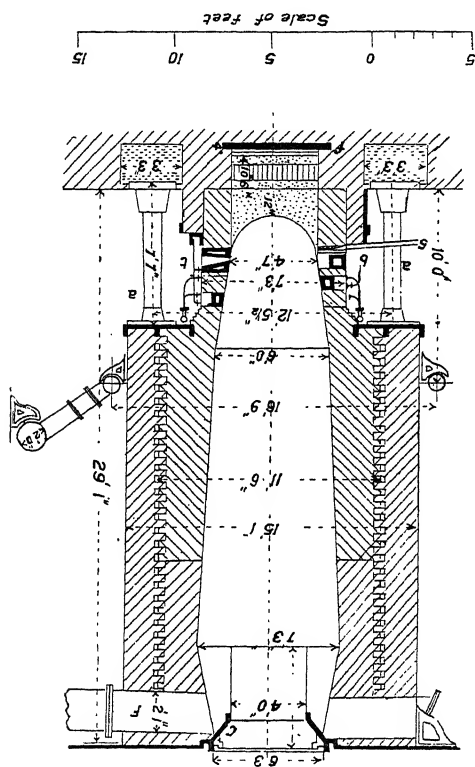
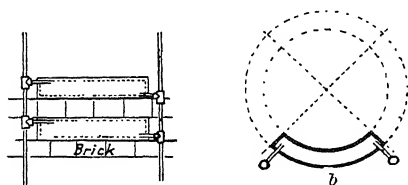


Fig. 80.—Przilbram Furnace.

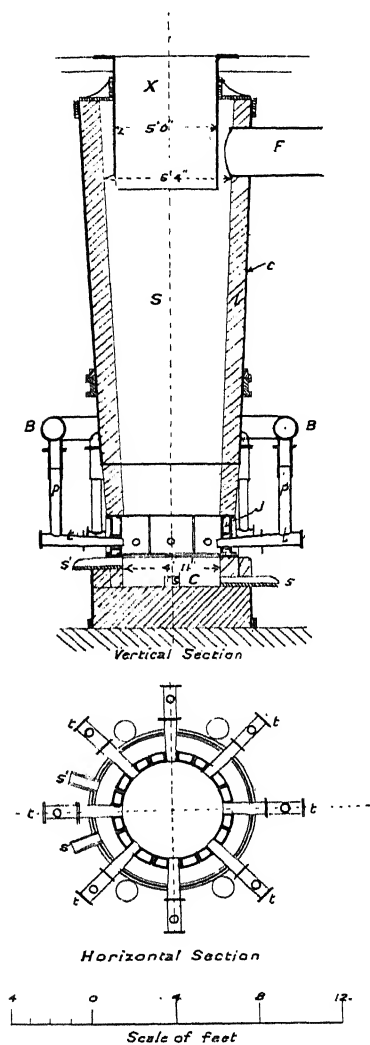


Figs. 80A and 80B.—Przilbram Furnace—Water Blocks and Bricks.

little improvement during the past twenty-five years or more, while in America it was soon improved out of all recognition by a series of able German and American metallurgists, backed by powerful and progressive capitalists.



The water-jacket of this furnace is of wrought iron, and is built up in segments, each segment having a tuyere aperture in the centre. The construction of the furnace is shown in Figs. 81 and 82. *S* is the shaft, *c* the casing of boiler-plate surrounding the brick lining, *l*, which rests on four cast-iron



Figs. 81 and 82.—Pilz Furnace.

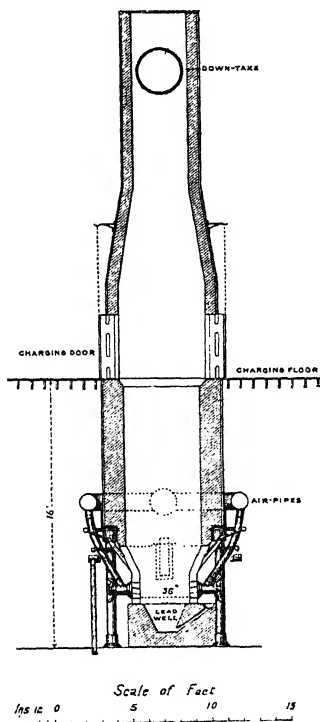


Fig. 83.—American Circular Water-jacket Furnace.

columns. *J* are sections of the jackets, each 8 inches wide and 20 inches high, with an iron tube through which are inserted the tuyere nozzles, *t*. Cold water enters through a small pipe opening into the bottom of each section, while the hot water is discharged through another pipe at the top.

The tuyeres are supplied with air by means of the sheet-iron bustle pipes, *p*, hanging from the cast-iron blast pipe, *B*, which surrounds the furnace. *C* is the crucible, lined with firebrick and furnished with two spouts, *s*, at the level of the bottom, for tapping lead and matte into the movable tapping box on wheels, and with others, *s*<sup>1</sup>, at a higher level for the slag. *F* is the flue for the escaping furnace gases, connected with the general dust chamber and flue system of the establishment; *X* is the charging cylinder hanging in the centre of the throat. The capacity of the furnace, with a pressure of  $\frac{7}{8}$  to 1 inch of mercury, is from 30 to 35 tons of ore per day, the coke consumption being 22 per cent. of the weight of ore. For further particulars of the furnace work *v. Chap. xi.*

6. **The American Circular Water-jacket.**—One variety of this furnace is shown in sectional elevation in Fig. 83, in which the charging floor and doors are plainly visible, as are also the blast pipes, tuyeres, and the outside lead well communicating by means of the automatic tap with the interior crucible. The whole shaft of the furnace is cased with iron plates, a continuation of which forms the stack, and the iron casing and stack, together with its lining, is supported on four heavy cast-iron columns. This particular furnace is fitted with the Devereux patent tuyeres, to be described in their place, and is charged from both sides, the flue-dust and waste gases being removed by an overhead downtake.

Circular furnaces were used at Eureka, Nevada, and, in the early days of water-jacket smelting, all over the United States. At *Mine La Motte* and other places in Missouri, furnaces more or less of this type are still in use. The diameter at the tuyeres is from 3 feet 9 inches to 4 feet 3 inches, the jackets are 3 feet 6 inches high, with a bosh of 3 inches, which is continued upwards to the charging floor 8 feet to 10 feet 6 inches above tuyere level. The crucible is 2 feet 6 inches deep, provided with automatic syphon tap for the lead, and the brick stack above the water-jackets has a boiler-plate casing, and is supported on columns in the usual way. There are five or six tuyeres supplied with air at a pressure of 1 to  $1\frac{1}{2}$  inches mercury.

Circular water-jackets are in use at Coueron\* and Pontgibaud† (France), at Mazzaron‡ and Puertollano§ (Spain), and at many places in Mexico and other countries. They are most suitable where labour is cheap and the ores not too siliceous. With large supplies of siliceous or refractory ores, poor in lead, they give way to the large oblong furnaces.

At Pertusola (Italy) and Peñarroya (Spain)|| large circular iron-encased brick furnaces with water-tuyeres and spray jackets are in use; for details of the work done see Chap. xi.

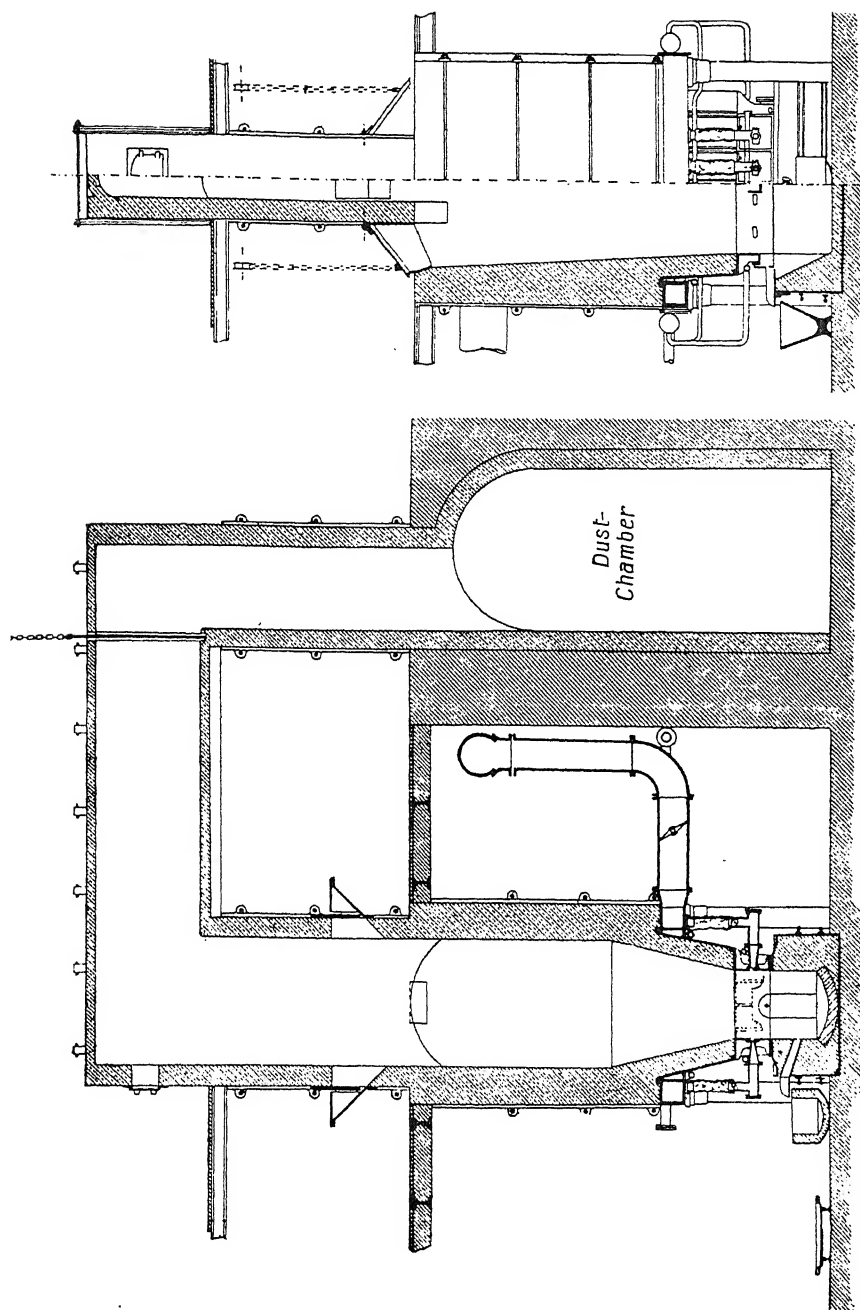
#### RECTANGULAR FURNACES.

The original oblong furnaces were of the Raschette type, first employed in the Urals for iron smelting, and subsequently in the Upper Harz for lead-silver smelting. These furnaces were not water-cooled; they were 16 feet 3 inches from tuyere level to throat, and measured 3 feet by 6 feet 6 inches

\* Philips, *Elements of Metallurgy*.

† ‡ § Sanchez y Massia, *Metallurgia del Plomo*, Madrid, 1893.

|| *Private Notes*, 1908.



Transverse Section.  
 Figs. 84 and 85.—Rectangular Furnaces with Spray Jackets (Monteponi).  
 Longitudinal Section (half elevation).

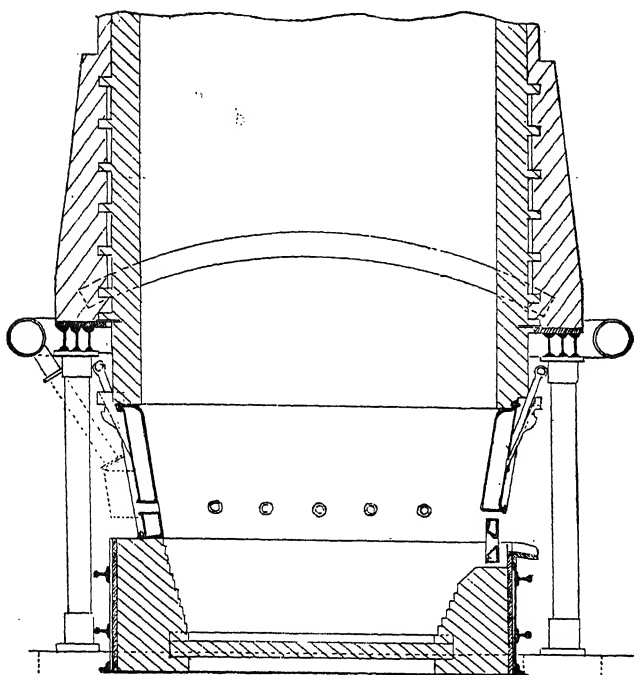
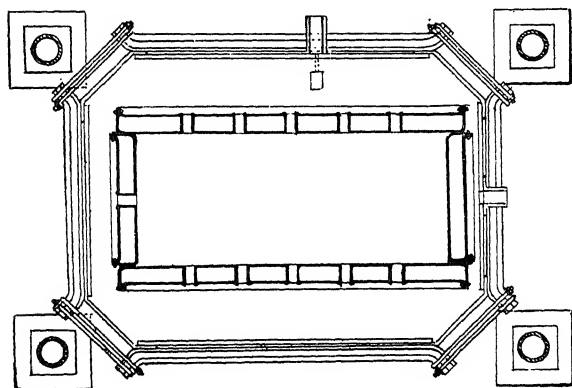


Fig. 86.—Rectangular Furnace—Sectional Elevation.



ins 12 6 0                      5                      12 Ft

Fig. 87.—Rectangular Furnace—Sectional Plan

at the tuyeres. On each long side were situated five 2-inch tuyeres; the hearth was saddle shaped, so that at each end there was a sump made of brasque, which was tapped at intervals into an outside tap-pit in order to facilitate separation of the melted products.

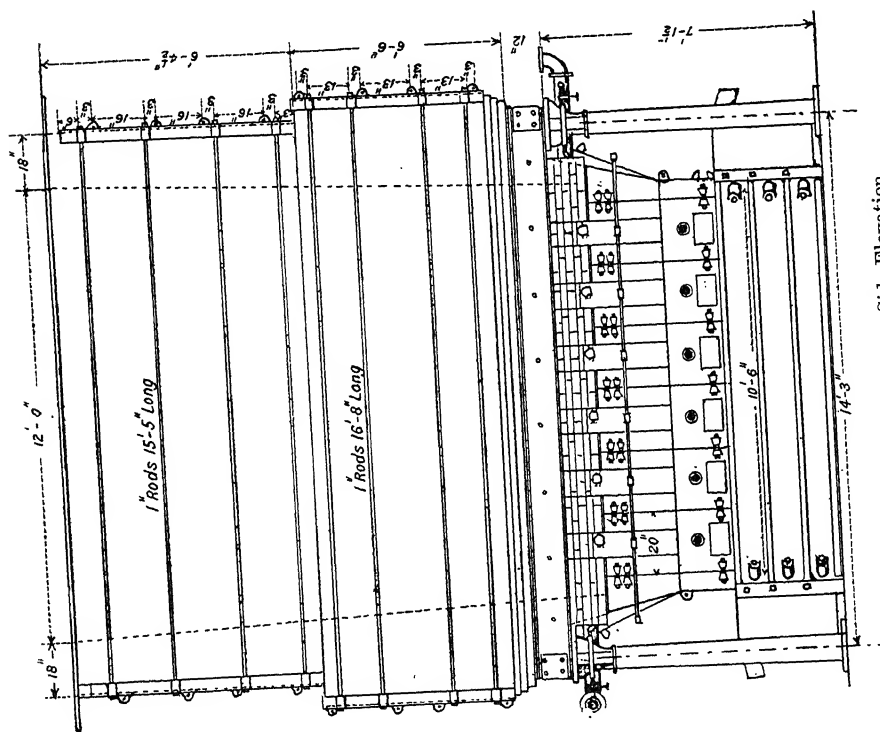
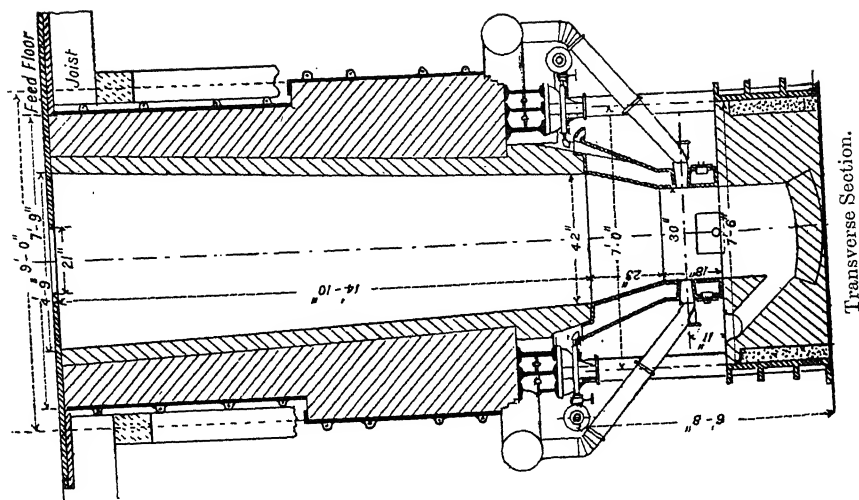
At *Monteponi* (Sardinia) the rectangular furnaces employed are a great advance upon the original Raschette furnaces, and are similar in some respects to modern American furnaces. As shown in Figs. 84 and 85,\* they have four tuyeres on each side, and instead of complete water-jackets they are furnished with spray-jackets. These consist of cast-steel plates 20 inches high, cooled by means of a water spray, and connected with the upper part of the furnace shaft by means of a bosh wall, composed of one thickness of firebrick surrounded by a boiler-plate bosh-jacket, which can also be cooled by means of a water spray. This arrangement demands much less water than would be required for proper water-jackets, inasmuch as part of the water used is vaporised, so utilising the latent heat of steam for cooling. There are only two patterns of jacket plates—namely, two side sections, each with four tuyere slits, and two end sections each with a circular hole 1 foot in diameter for a slag breast, closed by a thin sheet-iron plate with a taphole in it. When the breast needs attention, or access is required to the interior of the furnace, this sheet-iron plate is removed and the clay breast broken away. The upper part of the shaft is supported upon a rectangular hollow mantel frame of box section, which serves also as bustle-pipe. The crucible is provided with a syphon-tap for the lead. The upper works of the furnace above the feed floor are somewhat peculiar, as will be seen from the figures. Only the side doors are ordinarily used for charging from iron cars, the end doors covered by counterbalanced iron plates being seldom used except when barring down, which is frequently required, owing to the high percentage of zinc in the ores.

#### MODERN FURNACES AND ACCESSORIES.

In order to give an insight into furnace construction, it will be best to first give illustrations of a variety of typical furnaces, and then consider separately the detail construction of each of the important parts of a furnace as exemplified by the different types illustrated. Figs. 86 and 87 † show a furnace with wrought-iron jackets erected at *Great Falls* (Mont.) in 1891, Figs. 88 and 89 one of the furnaces with cast-iron jackets erected at the *Globe Works* (Denver) in the same year, while Fig. 90 shows the lower part of a furnace erected at the same works in 1895, and Figs. 91 and 92 show the furnaces of the B.H. Proprietary Block 14 Co. at Port Adelaide, South Australia, erected in 1897. To compare with these as examples of more recent furnaces, Figs. 93 and 94 show the new *Port Pirie* furnaces of the B.H. Proprietary Co. erected in 1906, and Figs. 95 to 100 show the new furnaces erected at *Laurium* (Greece) in 1908. Of these, Figs. 95 and 96 give an idea of the general arrangement of the furnace building, and Figs. 97 and 98 show the furnace shaft in relation to the hood, forehearth, and other accessories, while Figs. 99 and 100 give details of the furnace shaft itself.

\* Ferraris, *Oest. Zeits. f. B. u. H. W.*, 1905, p. 455, and Summary in *Eng. and Min. Journ.*, Oct. 28, 1905, p. 781.

† From Hofman, *Metalurgy of Lead*, 1906, p. 233.



Side Elevation.  
Figs. 88 and 89.—Globe Furnace (Denver), 1891.

**Dimensions.**—The sectional area of modern oblong furnaces has been increased during the past fifteen years by increasing the length, it being, as already pointed out, impracticable to widen these furnaces to above 5 feet at the outside, and even that width has only been considered practicable during the past few years, since the effect of blast-roasted material in loosening up the charge-column has been realised. Fifteen years ago there were no furnaces in existence over 12 feet long, and most of those in use were much smaller; none had an effective width between tuyeres of more than 3 feet 8 inches. To-day, 160 to 180 inches, or  $13\frac{1}{2}$  to 15 feet, is about a standard length, while there are a large number in blast which measure 16 to 18 feet. Similarly in regard to width, which had remained stationary for many years at about 42 to 46 inches, owing to the increase in the amount of fines treated,

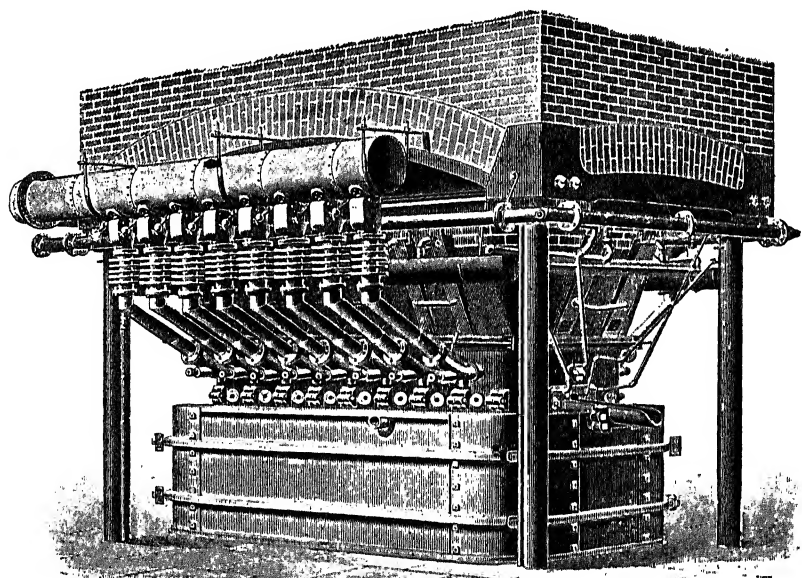
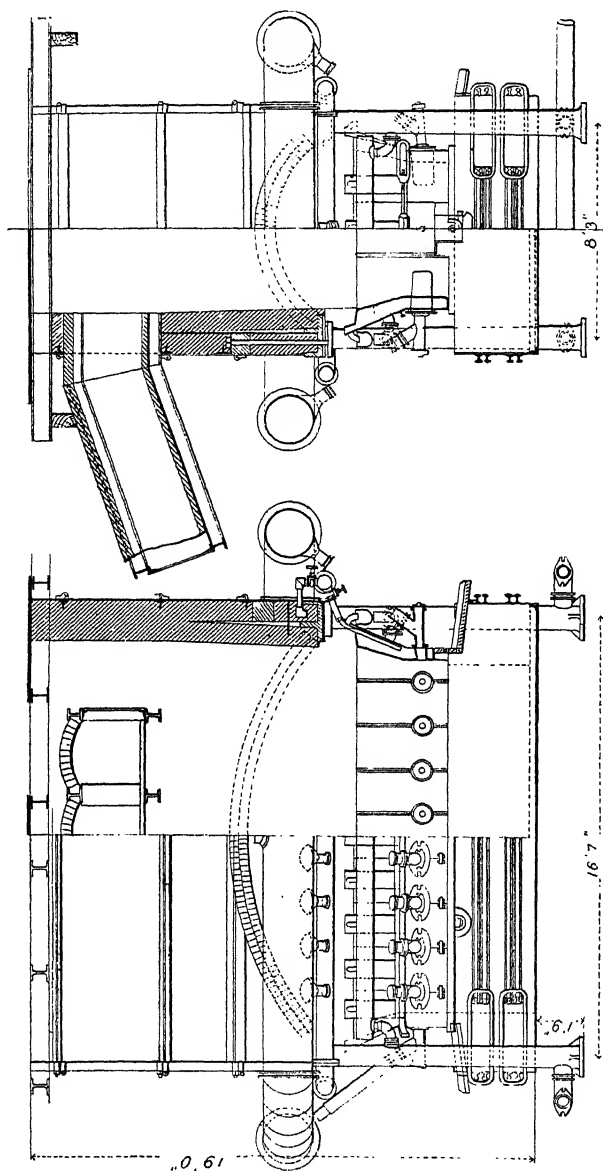


Fig. 90.—Globe Works (Denver), 1895.

the influence of higher blast pressures, coupled with the loosening effect of blast-roasted charge material, has been to increase this to from 48 to 60 inches. The *Port Pirie* furnaces figured above measure 62 inches between tuyeres, and at *Laurium* the installation is projected of a still newer furnace than those above figured, which shall be no less than 64 inches in width at tuyeres, the blast pressure required being estimated at  $3\frac{1}{2}$  lbs. per square inch.

The height of water-jacketed furnaces has been much increased of late years, in order to cope with more refractory ores, and to gain tonnage capacity while reducing dusting and volatilisation losses; and now averages something like 20 feet from tuyeres to stock line, being in some cases even greater, up to 24 feet.

1. **Foundation.**—A good foundation is essential, and solid rock is always best, unless more than 10 feet from surface; in which case two layers of timber



Figs. 91 and 92.—B.H. Block, 14 Furnaces, Port-Adelaide.

better, of old rails placed crosswise and followed by a block of concrete at least 4 feet, and, better, 5 feet thick may serve instead. The excavation



should be at least 3 feet larger each way than the furnace bed-plate, and is filled either with good rubble masonry, brickwork, or, better, by a block of

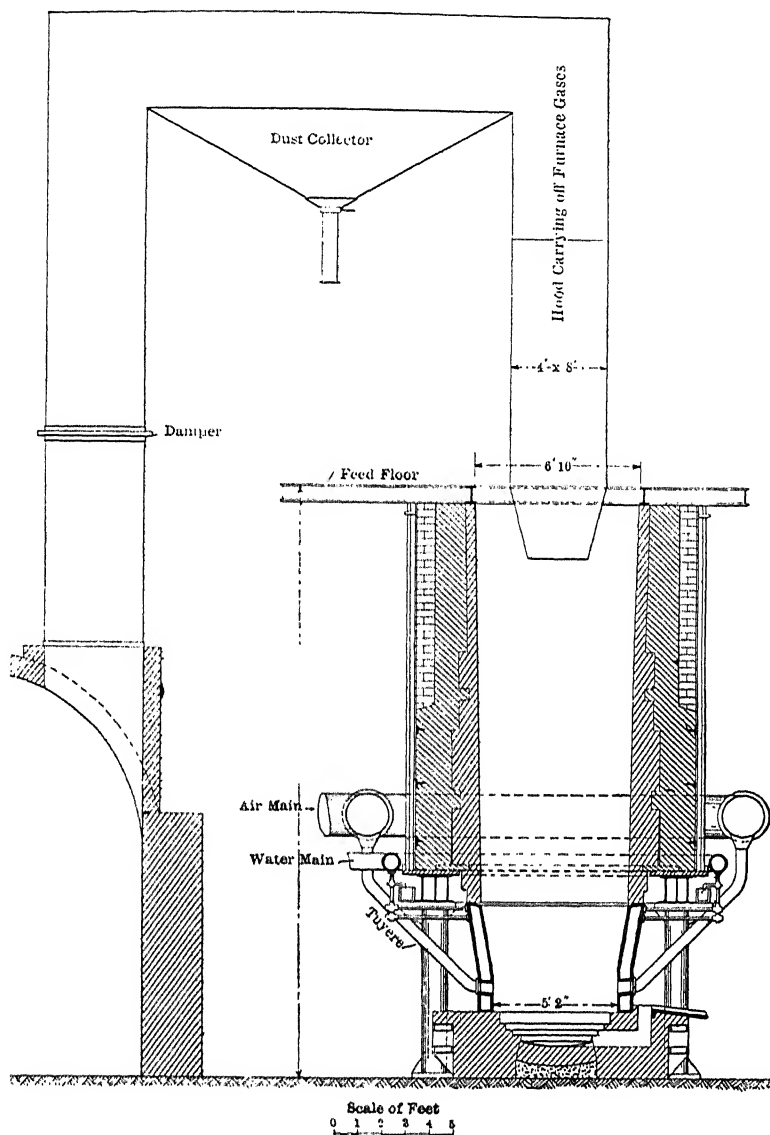


Fig. 93.—Port Pirie Furnace—Section.

concrete into which are rammed as many large stones as possible, cement being in any case mixed with the mortar. The best of all possible foundations is formed by filling the excavation with molten slag, forming, when

cold, a single block; but this is only attainable for additions to plant in an already existing works where a sufficiently quick succession of slag pots can be obtained. In any case, the topmost course must be perfectly smooth and level which, in a new works, is secured by making it of brick, and when molten slag is available can be easily obtained by forming shallow dams with pieces of bar iron (the tops of which are carefully levelled) and then

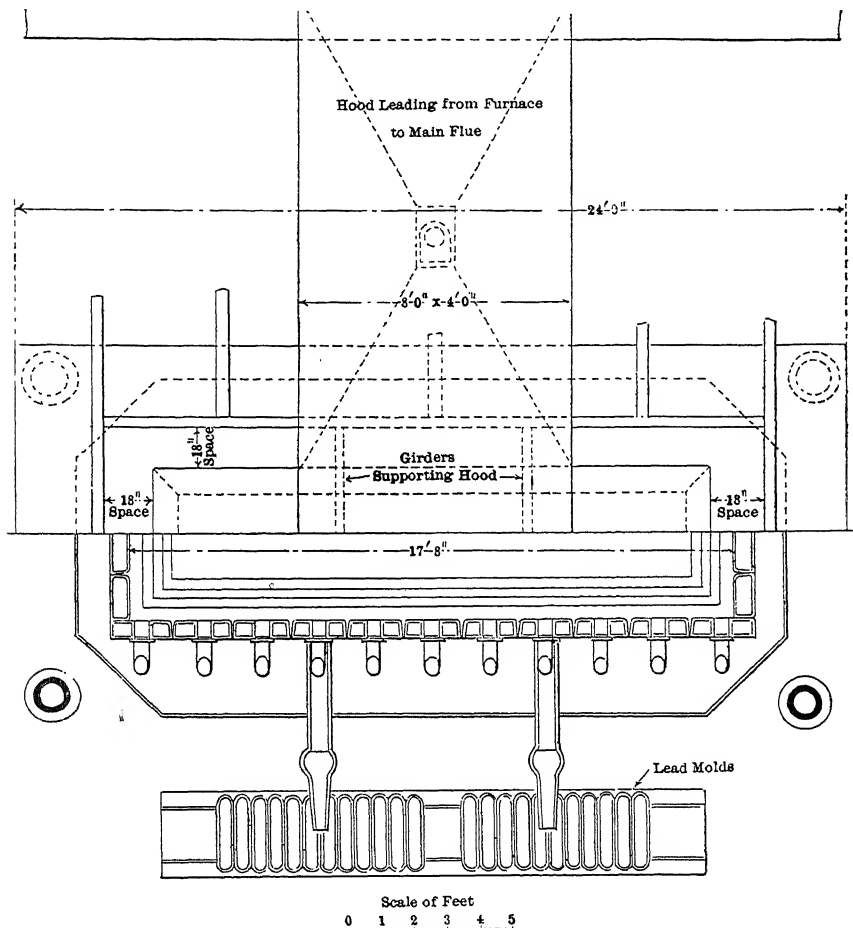


Fig. 94.—Port Pirie Furnace—Plan (part section).

filling them successively with molten slag. When a perfectly even and level foundation has been obtained, a layer of clay mortar is spread over it, and then the wrought-iron or steel "base plate,"  $\frac{1}{4}$  inch or  $\frac{3}{8}$  inch thick is laid in position. This plate has generally a rim of  $1\frac{1}{2}$ -inch angle-iron rivetted on to assist in holding the hearth castings or "caisson plates" in position, and its function is to prevent leakage of lead from the well or crucible.

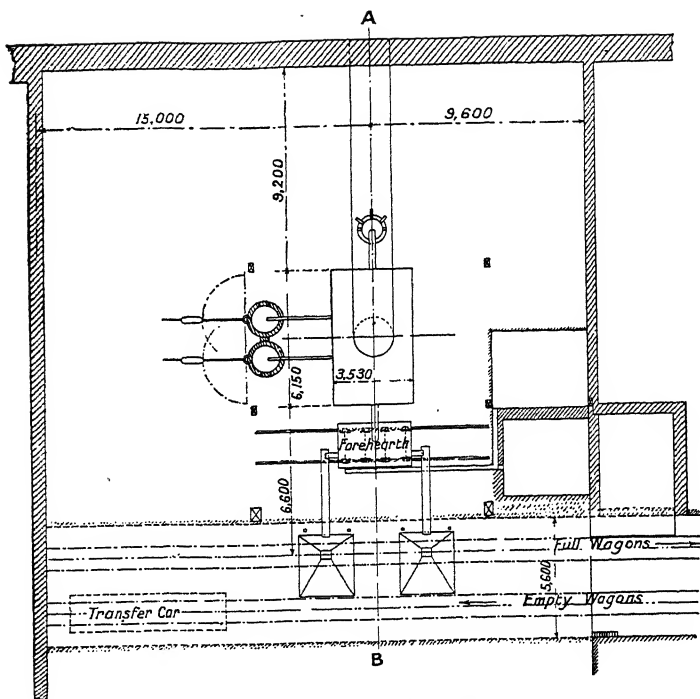
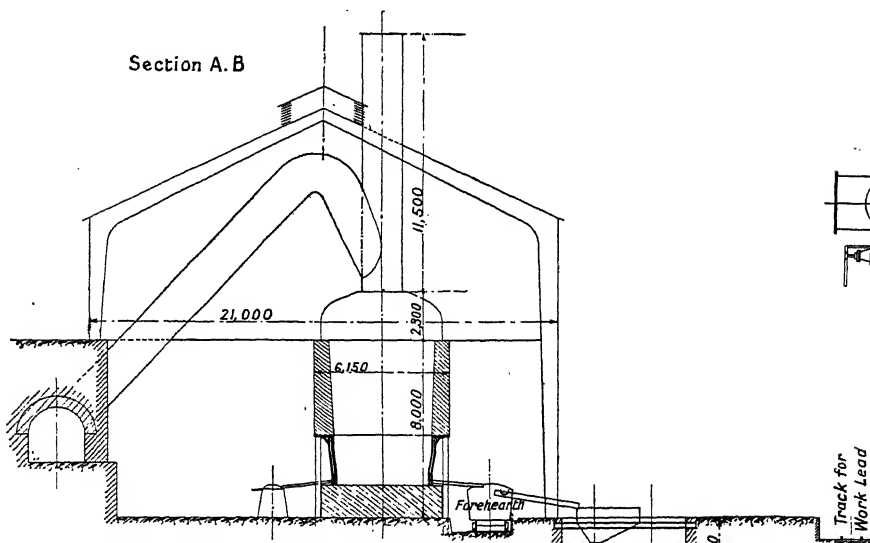
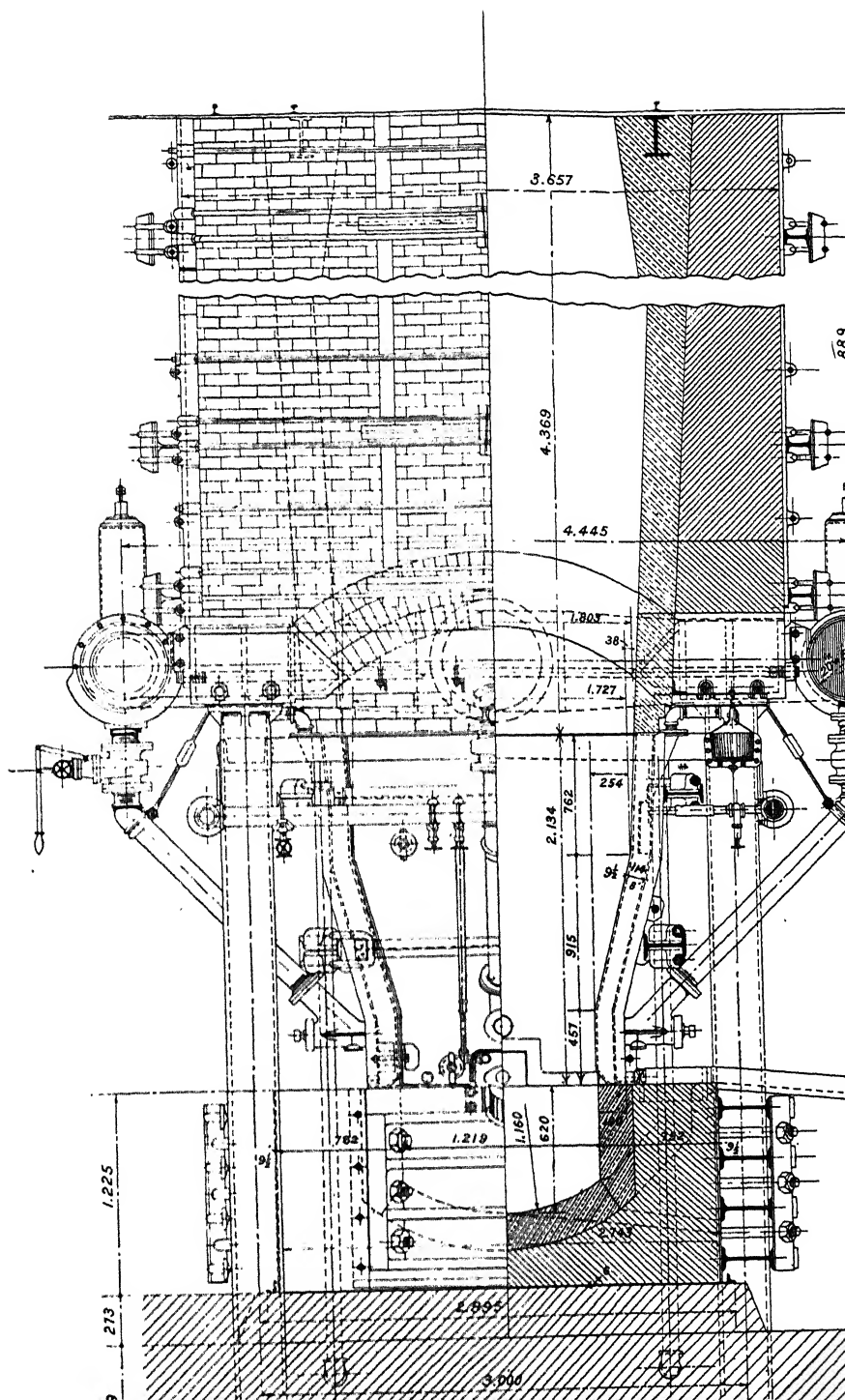


Fig. 95.—Plan (General).





2. **Crucible with Automatic Tap.**—The crucible of this furnace, built up on the bed plate, of firebrick laid as closely as possible with thin fireclay mortar, used to be surrounded by thick “crucible castings” or caisson plates, which had to be very heavy in order to resist the pressure of the lead in the crucible, frequently 30 to 50 tons dead weight of lead alone, without counting the downward pressure of the column of charge which floats upon it. These castings were commonly strengthened by projecting ribs and bolted together. Not only, however, did their great weight render them awkward for use at localities remote from railroad facilities, but they were always liable to crack through even slight subsidence or unequal distribution of pressure, and at most modern works both on the American continent and in Australia they have been replaced by strong I-beams surrounding a boiler plate casing, as shown in Figs. 97 and 98. Hofman, indeed, suggested \* that the outer casing of the crucible should be elliptical in form, when it could be enclosed in a boiler-plate shell strengthened by means of continuous bands of flat bar iron; the increased width necessary from centre to centre of the furnaces, which means a waste of space, and the increased length, which would have to be given to the syphon channels of the lead wells, thus increasing the liability to crucible troubles, are objections to this design which, so far, has only been adopted at the U.S. S. and R. Co.’s plant at *Bingham* (Utah).

Inside the plates the well is built up, a space about 2 inches wide being left between them and the firebrick, which is afterwards tamped with sand or brasque, so as to allow for expansion. The Arents automatic tap is simply an inclined channel half-a-brick square, which is left from the bottom of the crucible—generally at the centre of one side—to the outside, where a basin is formed called the “lead well.” Formerly this was formed by a semi-circular casting bolted to the crucible castings, but it is now wholly enclosed by the latter, and a spout is provided for the lead to flow into a cooling pot, which may be either fixed or, more generally, runs on wheels. In some old furnaces a lead well on each side was provided, but this is simply a waste of space, and although two lead wells are generally found on modern long furnaces, they are both on the same side. When the side instead of the end of the furnace was presented to the dump, two slag spouts were frequently found, one at either end of the furnace; since the ability to draw slag alternately from either end of the furnace or on alternate shifts is frequently of use in preventing the formation of crusts. On account of the convenience of handling charges on the feed floor, of the saving of space in furnace in furnace buildings, and for other reasons, it is, however, found convenient in most, if not all, modern plants to arrange the furnaces with their longer axes parallel to each other and transversely to the building—that is, with their ends facing the dump; and, in that case, since no permanent provision can be made for removing slag from the back ends of the furnaces, a slag spout at that end, if provided at all, can only be used during blowing-in or blowing-out, and in other cases of emergency.

Modern crucible construction is well exemplified by Fig. 101, which shows details of the crucible of the *Laurium* furnace. Inside the angle-iron rim of the mild steel base-plate are placed on edge the enclosing steel plates, firmly bolted together at the corners, and bound by no less than four 12-inch steel I-beams in order to take up the stresses and so prevent cracking and conse-

\* *Metallurgy of Lead*, N. Y., 1906, p. 236.

quent leakage. These are lined first with 11 inches of brasque, followed by a full course of ordinary firebrick, the bottom of which is laid as an inverted arch resting upon the brasque previously cut out to that shape. Inside the firebrick is a full course of special chrome-magnesia bricks made of a mixture of 3 parts of calcined magnesite to 1 of chromite, and burnt at the highest attainable temperature to prevent subsequent shrinkage. This mixture has been found to resist the corrosive action of the large amount of iron speiss made at these particular works better than any other. Two lead syphons 8 inches wide are provided, only one of which is in use at a time, and opposite the bottom of each on the other side of the crucible is an emergency taphole, which allows of emptying the crucible in case of necessity.

3. Shaft.—At each corner of the foundation stands a column (usually of

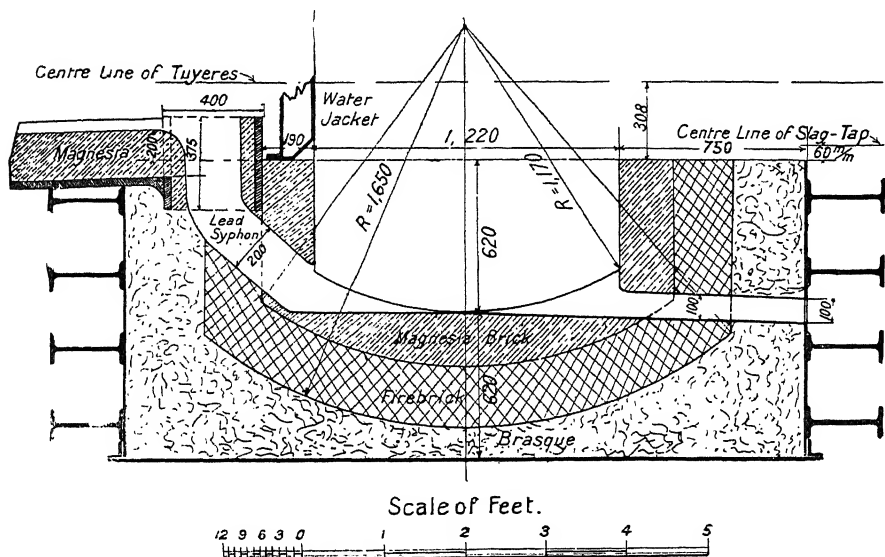


Fig. 101.—Crucible of Laurium Furnace. Dimensions in Millimetres.

hollow cast-iron) about 8 inches in diameter and 1 inch thickness of metal, with capital and base each 16 to 20 inches square and 2 inches thick; but often built up of wrought iron H-section. Upon these columns is set the "deck frame," which supports the whole of the furnace walls with their lining. In old furnaces this was composed simply of cast-iron plates  $1\frac{1}{2}$  to  $2\frac{1}{2}$  inches thick, with strengthening flanges 3 to 9 inches deep, part of the direct pressure being taken off by arches in the brickwork. These light plates, however, were always giving trouble by bending and breaking, owing to unequal expansion, and in all modern furnaces the deck frame is composed either of three I-beams bolted together supporting a plain flangeless cast-iron deck plate in two sections running the full length of the furnace, or of a strong brick arch sprung from heavy cast-iron skewbacks resting on the corner columns. With the former arrangement, shown in Figs. 86 and

89, the I-beams are bolted firmly to the caps of the columns, and to each other through distance pieces, while the cast-iron deck plates simply rest upon them so as not to transfer any of the lateral thrust caused by unequal expansion. If the arch arrangement is preferred, as is generally the case, the lateral thrust on these may either be relieved by supporting the brick arches throughout their whole length by bent steel arches, as shown in Fig. 90; or, a still better arrangement is to take up practically the whole of the thrust on the skewbacks by means of heavy tie-rods, and this plan (shown in Figs. 99 and 100) has become practically universal in recently built furnaces.

The furnace walls resting on the deck plates are usually of common brick with an inner lining of firebrick, an expansion space being sometimes left between the two layers, though it is difficult to see what useful purpose is served by it, and it is omitted in many recent furnaces. The walls of all modern furnaces are built much thicker at the bottom than at the top, which diminishes the loss by radiation. In some cases, where firebrick is obtainable at not more than double the cost of common brick, the whole stack has been built of firebrick throughout, which much lessens the liability to repairs.

The whole shaft is well braced with tie-rods and heavy corner irons of cast iron with lugs to receive the tie-rods. With the greater stresses on the high shafts nowadays employed, the ironing has to be much heavier than that which formerly sufficed, and it is usual, not only to employ strong I-beams both longitudinally and transversely at intervals, but to further support the brickwork by flat irons placed vertically at intervals inside the I-beams. Fig. 99 gives a good idea of the best way of ironing a stack.

In some furnaces, and notably in those designed for the British Broken Hill Company, a copy of which was erected in Tasmania, the whole upper part of the furnace shaft was made of wrought-iron and mild steel water jackets, as has been done with copper furnaces for many years past. This was expected to facilitate the "barring down" of zinc "hangings," and to give an indestructible shaft. In practice, however, these long side sections have been found to give considerable trouble by buckling, and their cooling effect is so great that there is a quite perceptible increase in the fuel consumption. The zinc hangings do not seem to separate so easily from the smooth iron surface as was expected, while the quantity of accretions which form is (owing no doubt to the powerful cooling effect) much greater than with the ordinary brick stack. The use of these upper jackets has not, therefore, been extended to other than Australian works.

**4. Charging and Fume-collecting Arrangements.**—One arrangement is to have an overhead stack and downtake for the furnace gases, built either of brick or of sheet iron, the charging being done through doors in each long side. This style is used at the Arkansas Valley (Leadville), Germania (Salt Lake City), Colorado Sm. Co. (Pueblo), also at Monterey, Aguascalientes, Laurium, and other plants; it answers very well for charges not too high in zinc, and where, consequently, there is not much "barring down" to be done. The other, and perhaps more common arrangement, is to feed the furnace from the top, drawing off the gases by means of a "take-off" below the feed-floor, as shown in Figs. 91 and 92; this system was given the preference at the Pueblo and Philadelphia Works (Pueblo), Globe (Denver), El Paso (Tex.), San Luis Potosi, Torreon, and all the Australian works, except the new Port Pirie; it is much more convenient in many respects,

but requires a greater height from the dump level, and, unless proper movable cover-doors are provided, a more powerful draught to prevent the escape of fumes on to the feed floor. It is, however, the most convenient, on the whole, for installing mechanical charging arrangements (for a description of which see the next chapter).

The feed-floor is made of cast-iron or wrought-iron plates with planed edges butted together, and wrought-iron cover-plates underneath, the bolts having countersunk heads on top. In order to avoid the exit of fumes on to the feed floor when open tops are employed, an old device was to provide each furnace with a telescopic ventilating hood made of sheet iron, consisting of a fixed chimney in the roof with a hood and smaller pipe sliding up and down in the former, and counterbalanced with weights so as to be easily moved up and down. These hoods have now been, in most cases, abandoned. Another arrangement employed formerly at some large works (including the Grant works, Denver, and the old El Paso works) is the *Pfort curtain*. This consists of an oblong or elliptical "thimble" of thick sheet-iron extending downwards from the charging floor as far as the level of the bottom of the "take off," and, at least, a couple of feet each way smaller than the furnace top. This appliance concentrates the down draught into a smaller space, and is found to trap the fumes effectually, but has the great disadvantage of being in the way when wall accretions have to be barred down, and, therefore, has been of late generally discarded, being especially unsuitable in works where charges high in zinc have to be treated.

As pointed out by Dwight,\* a still graver disadvantage of the *Pfort curtain* is that of accentuating the tendency of open-top furnaces to show a concentration of fine material in the centre, while the coarse fragments roll away to the sides, the evil consequences resulting from this arrangement of the charge components will be referred to in the next chapter.

The best arrangement for hand-feeding open-top furnaces is to allow the cast-iron floor plates to nearly cover the mouth of the furnace, except two spaces each about 4 feet by 2 feet, which are sufficient to enable the feeder to spread the charge in any desired fashion and to reach any part of the walls with a bar. Under ordinary circumstances these charging holes are left uncovered, but when blowing out, a cast-iron plate is placed over each opening and luted down with clay or wet ore-fines. The iron plates round these openings are not a permanent part of the feed floor, but are readily removable for barring down when required.

It is a matter of observation that furnaces with open tops require a much stronger chimney draught in order to prevent the escape of fumes on the feed floor than those which are run with closed top, and this extra draught always means extra loss. Some modern furnaces, therefore, are provided with the old fixed sheet-iron stack and telescopic hood in the roof overhead, connection with which is ordinarily made by means of a sheet-iron casing, the size of the furnace area, running on wheels and provided with side charging doors similarly to closed-top furnaces. When required, this casing can be run back for barring down.

The German plan of charging by means of cup and cone, as in iron blast furnaces, has been tried, but has not found favour, partly on account of the difficulty of obtaining free access to the furnace shaft for barring, &c., with

\* *Trans. A.I.M.E.*, vol. xxxix., p. 353.



such devices, which indeed is the principal objection as regards circular furnaces. For rectangular furnaces the proper distribution of the charge with any such arrangement would seem to offer insuperable difficulties.

During recent years somewhat elaborate arrangements for mechanical feeding have come into use on grounds both of economy and of reliability, it being found advisable, in order to secure regular results, to eliminate as far as possible the human element. Mechanical feeding is generally combined with open-top furnaces, the gases being drawn off below the feed floor. Details of the plant and arrangements adopted will be found described in the Chapter on Blast-furnace Practice.

At the new *Port Pirie* furnaces the unusual method of taking off the fumes by means of a central oblong *Darby tube* or hood, which hangs in the furnace to about 3 feet below the feed floor, gives satisfactory results. The open top of the furnace measures 19 feet 1 inch by 6 feet 10 inches, and the hood, made of plate iron, covers the whole of this space, except a space 18 inches wide all the way round, through which the buggies of charge are emptied into the furnace. In this way the whole of the coarse material runs to the centre, and the fumes naturally come up through the channel of least resistance.

A good deal of sickness (lead colic) used to be caused on the feed floor, as well as below, by fumes rising from the slag tap and lead well. The danger of lead poisoning is found to be minimised by completely partitioning off the feed floor from the lower part of the building, and providing the lower floor with its own series of shaft ventilators of sufficient size over each slag tap and lead well, and extending clear through the feed floor above the roof. They take up but little room on the feed floor, and ventilate the lower floor very effectually.

With ordinary open-top furnaces there is always more or less trouble at times on the feed floor from fumes, unless the tops are closed between charges by tightly fitting doors, which serve the purpose of confining the indraught of air exclusively to the set of doors open at any given moment, and so making the indraught stronger. It is one of the minor advantages of bag-filtering methods for saving flue-dust (a description of which will be found in Chap. xii.) that the escape of fumes at the furnace is reduced to quite small dimensions, and thus not only is a nuisance done away with, but the lead in the fumes is recovered.

5. **Water-Jackets.**—These may be either of cast or of wrought iron (in the most modern practice the latter are usually preferred), but in any case their usual height is from  $4\frac{1}{2}$  to 7 feet, the centre of the tuyeres being 10 inches to a foot from the bottom and the “bosh” of 6 to 10 inches, commencing from 8 inches to a foot higher up, to be continued in the brickwork. It will be borne in mind that the jackets have no weight to support, except that of the few layers of brick between their upper surface and the “deck plates”; they need not, therefore, be very heavy; in fact, they stand best when made as light as possible. They are almost invariably made with a bosh, but at the *Aurora Works* (Chicago, Ills.), *Eurich*,\* until only a few years ago, used a straight jacket and inserted water tuyeres 12 inches into the furnace to narrow the fusion zone to the required extent, the advantages claimed being the greater facility of barring off accretions, and a

\* *Private communication*, 1896.

small but noticeable saving of coke, owing to the fusion zone being kept at a greater distance from the jackets.

**Cast-iron Jackets.**—These are generally from  $4\frac{1}{2}$  to 6 inches inside measurement, the latter size being, perhaps, most common; and the iron is from  $\frac{3}{8}$  to  $\frac{1}{2}$  inch thick,  $\frac{5}{8}$  inch being, perhaps, the commonest; frequently the inner skin is made double the thickness of the outer, as formerly at *Broken Hill* (Fig. 103). Any difference of thickness, however, must necessarily cause strains due to unequal expansion and contraction, and it is, therefore, decidedly preferable to aim at having the iron of exactly the same thickness all round the jacket, as is indeed the common practice with cast jackets nowadays.

The water feed may be through a pipe screwed into the outer skin (Fig. 103), or through an open feeder forming part of the casting (Fig. 102); in either case, sufficient head must be employed to ensure the jacket being always full of water, which is secured by allowing the hot-water exit to rise some inches above the top of the jacket. The complete casing of a 120-inch furnace will

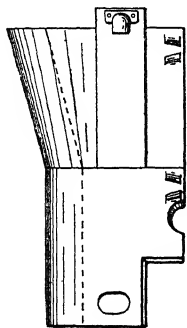


Fig. 102.  
Cast-iron End Water  
Jacket (curved  
type).

be usually made up of sixteen sections, in twelve of which the side jackets are exactly alike, each having lugs with which to bolt it to its neighbours and a tuyere hole cast in the centre. The front and back breast-jackets are each in two pieces, right and left, and in most modern furnaces are unprovided with tuyeres. In some cases they are made curved, and they never extend down to the top of the crucible, but stop some 10 inches above it, a common shape being that shown in Fig. 102, where the curved end-jackets extend right down at the sides, but leave an open space in the centre. The remaining space may be closed simply by pillars of firebrick with a clay slag tap in the middle, or by a tapping jacket, as is the more modern practice. The best arrangement is to have a tapping jacket 14 inches high, and to sink it 4 inches into the crucible wall, which helps to prevent leakage of lead from the front of the furnace, the actual slag tap being a conical hole in the jacket, only  $2\frac{1}{2}$  inches diameter on the outside, widening to 5 inches inside. In a small works, however, where the ore mixture varies widely in composition, especially if the ores be at all zinciferous, it is best to work with a firebrick breast instead of a tapping jacket, as it is then easier to get at crusts which form on top of the lead. Cast-iron jackets are always supplied with handholes for cleaning, and the feed-water has been, in one case at least, introduced through this handhole, which, however, can hardly be considered an improvement.

Detailed sections of cast-iron jackets are shown in Figs. 103 \* and 104,† the latter showing a novel arrangement for supporting each jacket separately and independently of the lugs which fasten it to its neighbours, the former being peculiar in its use of water-tuyeres, which nowadays seem to be restricted to certain Australian plants.

\* Broken Hill, 1896, *Private Notes*.

† Beardsley, *Trans. A.I.M.E.*, vol. xxi., p. 515.

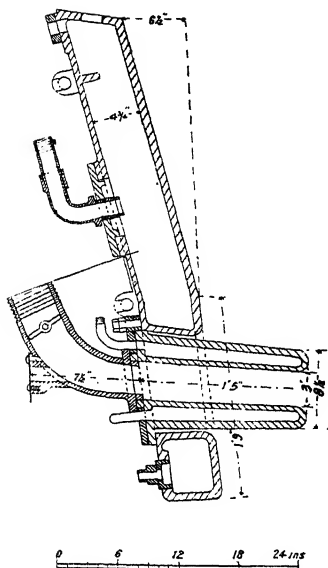


Fig. 103.—Section of Cast-iron Water-Jacket (Broken Hill).

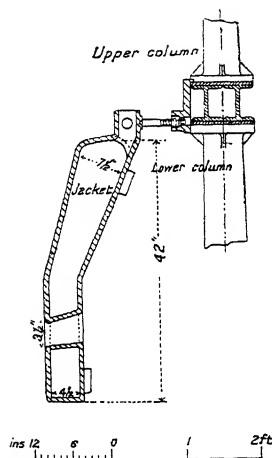


Fig. 104.—Section of Cast-iron Water-Jacket.

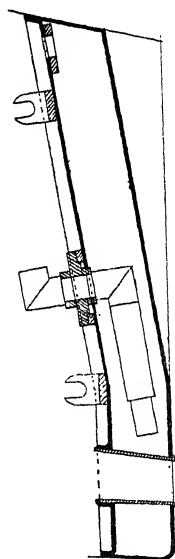


Fig. 105.—Section of Wrought-iron Water-Jacket (British Broken Hill).

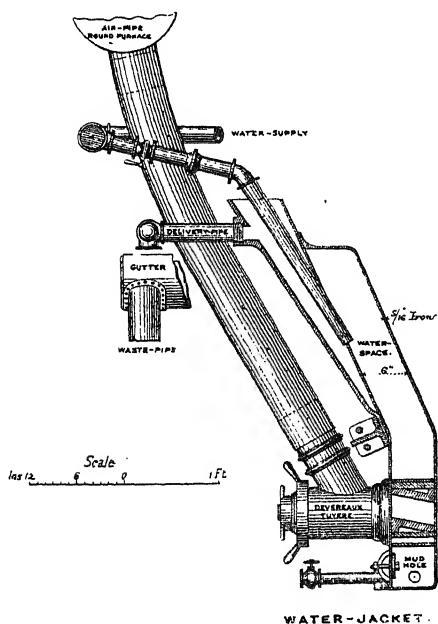


Fig. 106.—Section of Wrought-iron Water-Jacket with Devereux Tuyere.

Terhune highly recommended jackets of cast steel, which were in use for many years at the Hanauer Works (Salt Lake City). The sections are  $\frac{5}{8}$  inch thick, tapped for two bushings  $1\frac{1}{2}$  inches diameter for cleaning purposes, the same holes serving for water-feed and discharge by means of  $\frac{3}{4}$ -inch pipes in the plugs. Such jackets are, however, expensive, and do not seem to have come into use elsewhere.

**Wrought-iron jackets** are usually made somewhat higher than those of cast iron, but a smaller number of sections are required—viz., one for each end and three or four for each side, according to the length of the furnace. The old form of wrought-iron jacket had a feeder bolted on similar to that of the cast-iron jackets (Fig. 106). Another form is that of the British Broken Hill furnaces (Fig. 105), which has wrought-iron pipes screwed in

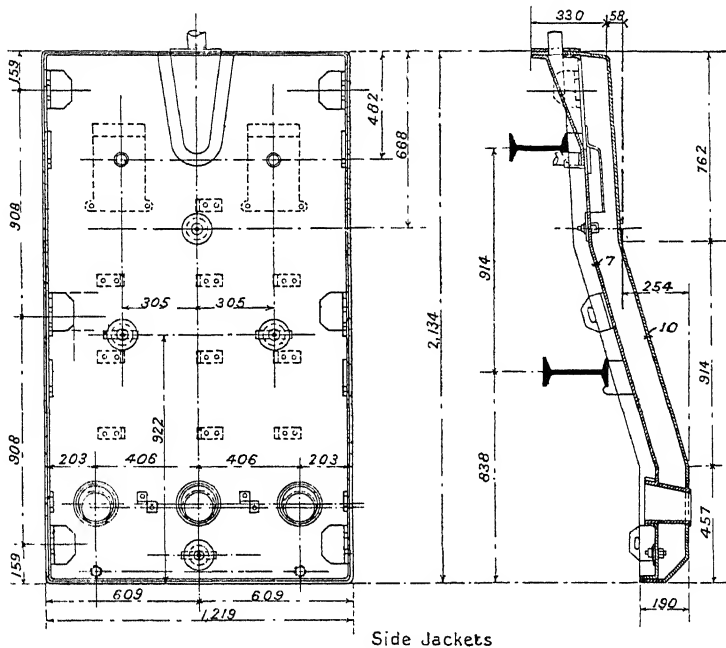


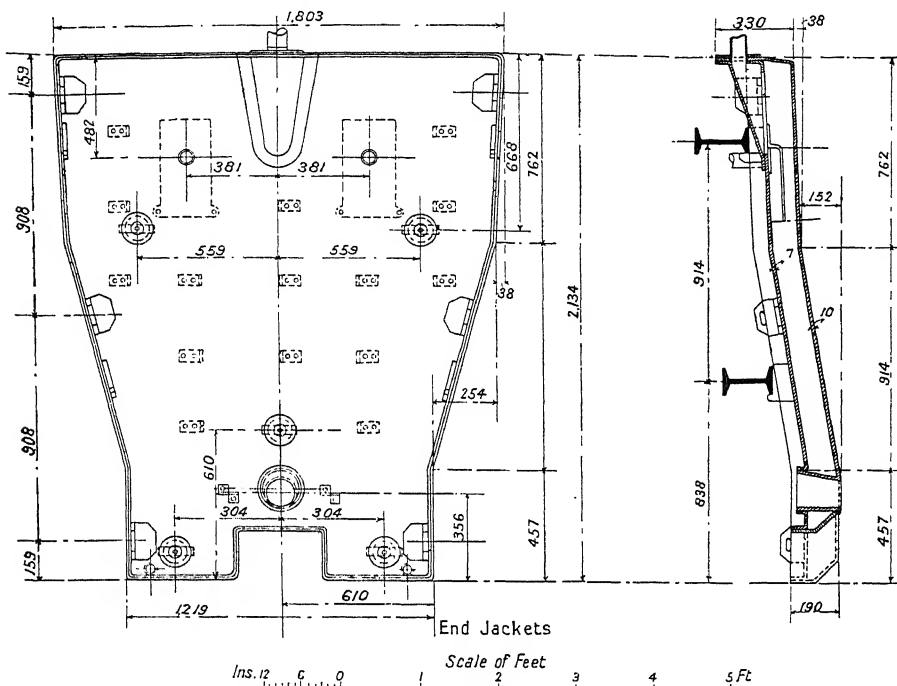
Fig. 107.

Fig. 108.

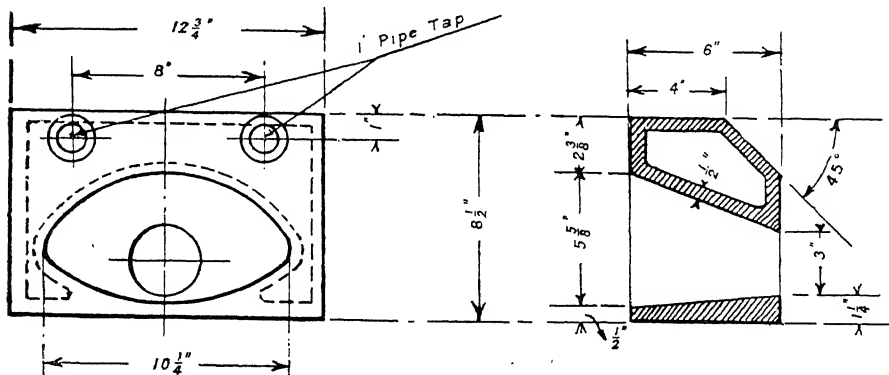
for both the feed-water inlet and outlet, and has lugs rivetted on for bolting each jacket to its neighbours. In any case, each section is provided with one or more handholes for cleaning out scale or sediment, and the walls of the long side-sections at any rate are stiffened by stay bolts.

Figs. 107 to 110 show elevations and sections of a standard type of jacket now much used, which has, besides lugs for bolting together adjoining jackets, angles for supporting two pairs of horizontal I-beams, which run respectively lengthwise and endwise, being bolted together by standard connection angles, and so take up the bursting stress of the charge. These particular jackets, too, are provided with deflectors to the inlet pipes, whereby the stream of cold water is forced to take a downward course, following the outer skin of the jackets, and they are provided with four or five handholes to a jacket.

The reason for the unusual number of handholes in this case is the use of salt water, but it is found that when the temperature of the effluent does not



Figs. 107 to 110.—Details of Wrought-iron Jackets with supporting I-Beams.



Figs. 111 and 112.—Cast-iron Tapping Jacket (Colorado Ironworks pattern).

exceed  $60^{\circ}$  C. there is no danger of saline deposits. The dimensions given in the figures are in millimetres.

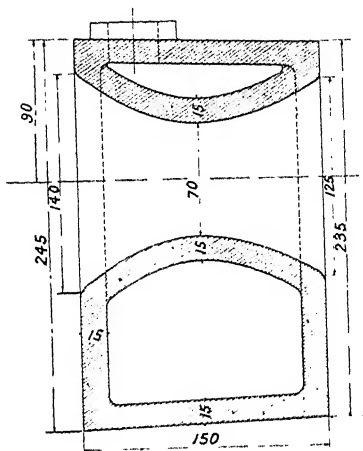


Fig. 113.—Section.

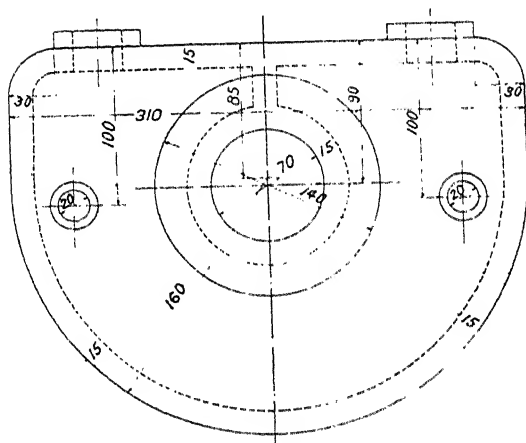


Fig. 114.—Elevation.

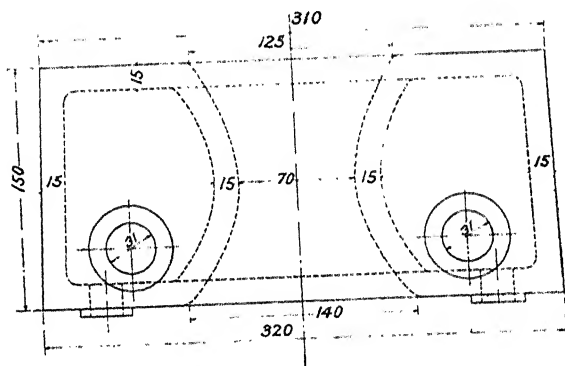


Fig. 115.—Plan.

Figs. 113 to 115.—Tapping Jacket (Laurium).

Wrought-iron jackets are best made of two sheets pressed out hot between suitable dies by hydraulic pressure, but they can be made by bending the sheets at a red heat over cast-iron moulds with hammers, the corners being carefully welded. A thicker sheet is usually employed for the inside than for the outside, common sizes being  $\frac{3}{8}$  or  $\frac{7}{16}$  inch inside, and  $\frac{5}{16}$  or  $\frac{3}{4}$  inch outside. The elevation of the end jacket (Figs. 109 and 111) shows the shape and size of the tapping jacket, detail of which is given in Figs. 113 to 115.

The life of cast-iron jackets, if well proportioned and made of good quality close-grained grey iron, is considerable, especially in large furnaces with regular running; thus, according to Croll,\* one of the 40-inch  $\times$  120-inch furnaces at the *Globe Works* having eighteen jackets ran continuously for fourteen months without changing a single jacket.

Comparing cast iron, therefore, with wrought iron, it should be remarked that within reasonable distance of a foundry which has had experience in casting them, the former are cheaper; for not only do they last nearly as long when properly made, and cost less, but, when replaced, the value of the old metal is a considerable discount off the price of new jackets; whereas a discarded wrought-iron section is of no value as scrap. Cast-iron jackets are especially to be recommended where the water is unusually hard or acid. Where, however, good foundries are not accessible, and the cost of carriage is high, the wrought-iron jackets generally turn out to be cheaper in the long run, though the first cost may be higher. Cast iron, too, is not so well adapted as wrought iron to the manufacture of the very high jackets used in modern plants.

Under special circumstances, as, for example, in places where wrought-iron jackets would have to be carried several thousand miles and so become costly, while common castings can be obtained locally, jackets may be made, as, for example, at *Antofagasta*,† of lengths of 1-inch gas pipe arranged in parallel rows joined up with elbows and nipples, the whole being placed in a mould to be cast into a thick plate. The quantity of water required for circulation through these plates in order to keep them cool is less than is required by the ordinary form of jacket, and although renewals are more frequent, the cost of renewal is less.

In a cast-iron jacket the tuyere-socket forms an integral part of the jacket itself. In the case of wrought-iron jackets the tuyere-sockets have to be fitted separately, and the usual mode of fitting is that shown in Fig. 116. A tapered hole having been cut in the inner skin of the jacket a little smaller than the outside diameter of the tuyere, the latter is turned off so as to leave a shoulder; the inner end of the tuyere is then expanded against the jacket and beaded over after the fashion of a boiler tube. The resulting joints are liable to leak more or less even in regular work, and are sure to do so if the tuyeres often get slagged, under the shock of the hammering required to clear them. The best way of fastening tuyeres on the inner skin is by autogenous soldering, or welding by means of the oxyacetylene blowpipe. Fig. 117 shows the method of preparing the joint for welding which is employed at *Laurium*.‡ The resulting welded joints never leak, and have given every satisfaction.

With the higher charge columns and blast pressures of modern practice, the smelting zone extends further upward, and it becomes necessary to carry

\* *E. and M. J.*, May 28, 1898, p. 639.

† *E. and M. J.*, Sept. 2, 1899, p. 279.

‡ *Private communication*, 1908.

up water-jackets to a greater height than formerly. Instead of having two tiers of jackets, however, as is a common practice in copper smelting, it is found desirable to carry the water-cooling to the required height in one single jacket, buckling being obviated and the necessary stiffness gained by making

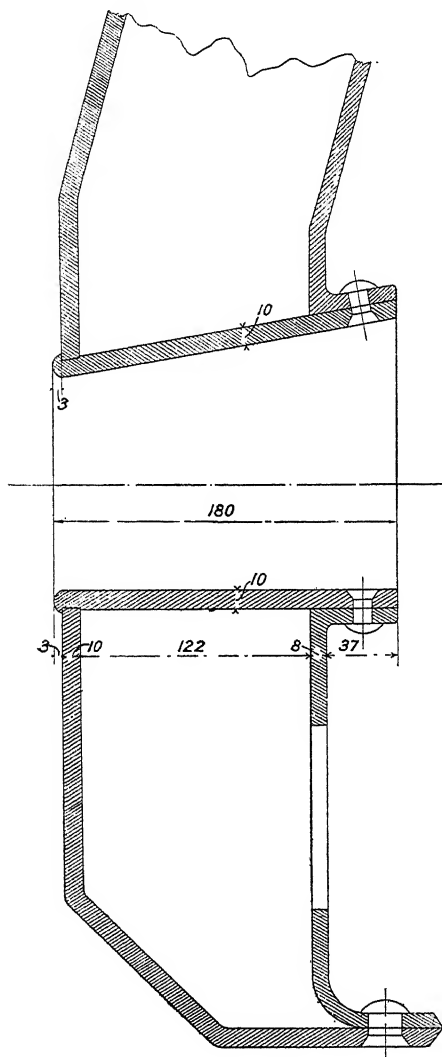


Fig. 116.—Beaded Tuyere.

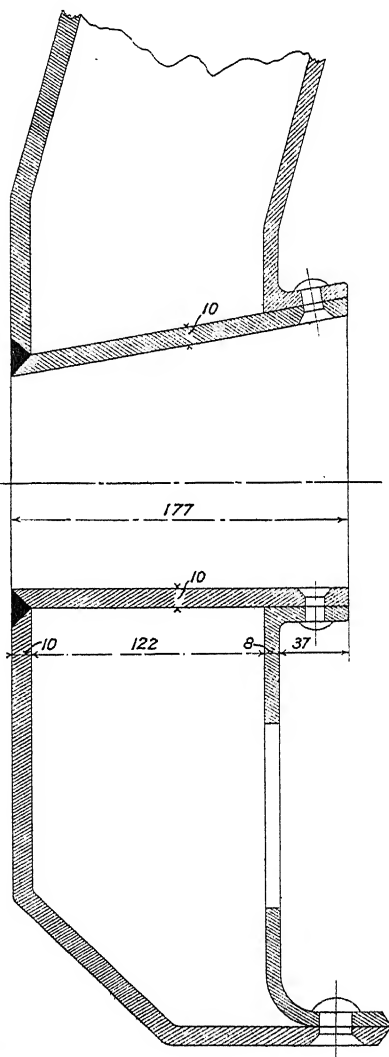


Fig. 117.—Welded Tuyere.

the individual sections narrower. The difference will be noted from a comparison of Fig. 99, which represents a furnace erected in 1908, with Fig. 86, which shows a low column furnace erected in 1891.



**Water-Jacketing to the Throat.**—Copper furnaces being frequently water-jacketed right up to the feed floor, the same principle has been tried in lead smelting, though with unsatisfactory results. The first furnaces of this kind erected were those of the *British Broken Hill Proprietary Company*, which are 40 by 120 inches at the tuyeres, the lower jackets being of cast iron and 20 in number—viz., 12 side jackets with tuyeres and 4 end jackets without (like Fig. 103). The upper portion of the furnace shaft is composed of four mild steel jackets, having lugs rivetted on them by means of which they are supported by an I girder frame resting on hollow columns, which form a continuation of the ordinary lower columns.

The advantages of water-jacketing the upper portion of a lead furnace were supposed to be:—(1) Greater durability through avoidance of relining; and (2) greater facility for barring off accretions. It was found, however, at *Broken Hill* that, through cooling of the upper portion of the furnace, the fuel consumption was slightly, but distinctly, increased, while the growth of “hangings” was actually encouraged by the presence of a cold surface to which they could attach themselves. The smoothness of the walls does somewhat facilitate the settling down of hangings, but not so much as might be expected; for, in any case, the “ring” of accretions has to be cut clean through in one or two places with steel bars before it can be got to settle down or break up, and this is the hardest part of the work. The upper jackets, moreover, were found to give considerable trouble through buckling, owing to their large size, and through leakage at the stay bolts; they have now been entirely abandoned. The furnace erected by Beardsley \* at Zeehan, Tasmania, was practically a copy of the British Broken Hill furnaces. In a few cases where short jackets have been employed with a high-charge column, the layers of brick between the water-jacket and the shaft proper have been cooled by means of a 1½-inch water pipe. This arrangement was tried in the *Germania* furnaces of 1895,† but has not come into general use.

**Tapping Jackets.**—These may be of wrought iron or cast, cast steel or bronze. They are sometimes built, as in copper smelting, of wrought iron, and as much as 18 inches square, but are often quite small. Figs. 111 and 112 show one of the Colorado Ironworks’ patterns of cast-iron tapping jackets which has been somewhat largely used in Colorado; not being completely water-cooled all round, this pattern cuts out rather rapidly, especially when much matte is made.

Figs. 113 to 115 ‡ show the pattern evolved at *Laurium*, which is cast in high-grade phosphor-bronze containing 98 per cent. Cu. and has a 2½-inch aperture. These jackets, when well cast without flaws, are found to last well under the hardest usage; and will withstand the passage of 200 tons of slag daily for an indefinite period, even when much matte and speiss accompany the slag.

**Water Supply.**—Details of the water supply to furnaces are indicated in Figs. 88, 89, 99, and 100; the cold water feed pipe, which may be from 3½ to 5 inches in diameter, according to the size of the furnace and the head available, surrounds the furnace under the blast pipe. Each jacket has a branch of its own about the middle or near the top, by which the water

\* *Trans. A. I. M. E.*, vol. xxi., p. 575.

† *E. and M. J.*, vol. lxx., p. 641.

‡ *Private communication*, L. Guillaume, 1908.

enters and sinks to the bottom, displacing the hot water which overflows through a spout into a galvanised trough surrounding the furnace that discharges into two or more stand pipes at the corners. These spouts and troughs are often replaced by long pieces of gas pipe reaching to the stand pipes near the corner columns, which latter indeed are themselves frequently made to serve also the functions of stand pipes (Figs. 91 and 92). In recent practice all the jackets are frequently connected at the bottom by short 1-inch nipples joined by thick rubber hose, or, better, by unions, so as to enable any one jacket to be fed from its neighbours in case of temporary failure of its own proper water supply. Deflectors are usually attached to each inlet pipe, as shown in Figs. 108 and 110, so as to prevent the stream of cold water from impinging directly upon the hot inner sheet of the jacket.

As to the quantity of water required, an ordinary furnace of 120 by 42 inches will take, in summer time, about 55 gallons per minute, supposing the water to be run into shallow ponds to be used over again while still warm, as is frequently the case. It is economical both as regards fuel consumption and wear and tear of jackets to keep the water supply to the jackets low, so that the escaping water shall be as hot as possible without generating steam.

It may be reckoned that in normal work a furnace with jackets 3 feet 6 inches high will require as a minimum from  $1\frac{1}{2}$  to 2 gallons per minute, according to the temperature of the water, for every foot of internal perimeter at the tuyere level, and double this amount must be allowed for blowing in or out. Furnaces with higher jackets will, of course, require more water, but not nearly in proportion to the increased height.

**Heat Loss in Jacket Water.**—It has been generally reckoned that about one-tenth of the heat generated by the combustion of the fuel is lost in the jacket water. Some experiments on the point by Austin \* may be quoted. At *Patagonia* (Ariz.) a furnace  $36 \times 120$  inches was found to take 2,000 gallons of water per hour, which, entering at  $22^{\circ}$  C. and leaving at  $67^{\circ}$  C., absorbed 750,000 calories per hour. The coke burned was 1,000 lbs. per hour, which at 7,000 calories per lb. = 7,000,000 calories. The heat absorbed by the jacket water is therefore, in this case, 10.7 per cent. of the total.

**Tuyeres.**—The diameter of tuyeres usually varies from 3 up to  $3\frac{1}{2}$  inches. The ratio between the number of square inches of total tuyere area and the number of square feet of effective sectional area at the tuyeres is conveniently termed the "tuyere-ratio" of the furnace. According to Hahn's rule, one 3-inch tuyere is required for each square foot of sectional area at the tuyeres, which is equivalent to a tuyere-ratio of 3.53. It will be evident, however, that for a given ore charge the tonnage put through will depend upon the quantity of fuel burned before the tuyeres in a given time, and this further depends upon the volume and pressure of the blast supplied, the latter again being dependent upon the height of the charge column. The actual tuyere area through which the given volume of air passes at a given pressure is only of importance as determining the velocity of the blast at the moment of entering the furnace, which affects the distance to which it penetrates the charge. We see, therefore, that for comparatively narrow furnaces the tuyeres may well be of somewhat large dimensions, whereas in wider furnaces it is advisable to use narrow tuyeres, in order to get good penetration to the centre.

\* *Min. and Scientific Press*, Oct. 17, 1908, p. 525.

In any case, good distribution of the blast demands that the tuyeres should be fairly close together, and this is facilitated by the use of wrought-iron jackets. Formerly with cast-iron jackets tuyeres were seldom less than 20 inches apart from centre to centre, and often more, whereas it is now common to find them 15 or 16 inches apart.

Tuyere nozzles are sometimes of bronze or brass (though now commonly of cast iron), bored and turned to fit as closely as possible into the conical holes in the jackets, which should be bored out to fit. In the old form of tuyere the bronze nozzle is rivetted on to a galvanised or sheet-iron elbow, and connection is made to this from the nipples on the branch blast pipe by means of canvas bags, soaked in alum or water glass to render them incombustible and air tight, and tightly wired on. In more modern forms of tuyere and with high-pressure blast, connection with the bustle pipe is effected by means of air-tight sheet-iron pipes provided with suitable flange connections. A peep hole at the back of the elbow is provided with a glass or mica eye-piece which can be removed when it is necessary to "rod" the tuyeres. The construction of the old form of tuyere will be seen by reference to

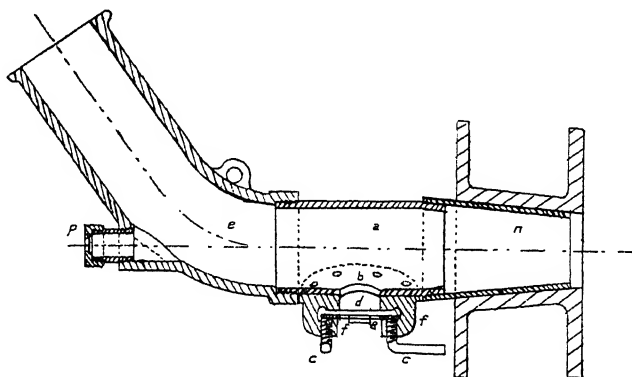


Fig. 118.—Ordinary Tuyere (Old Type).

Fig. 118, which is of the ordinary type formerly in common use, and in which *e* is the cast-iron elbow, *p* the peephole, and *n* the brass tuyere nozzle, the pipe with attachments between *e* and *n* being the slag Davies escape, a contrivance devised to obviate the bad effects of slag overflowing into the tuyere pipes, a not uncommon accident due to carelessness of the tapper. It consists of a piece of 4-inch pipe, *a*, screwed into the cast-iron elbow and turned so as to fit neatly into the brass nozzle; on its under side is rivetted the flange, *b*, with lugs, *f*. These hold, by means of screws, *c*, the perforated sheet iron, *e*, against the perforation in the bottom of the pipe, the hole being closed by the diaphragm, *d*, which is merely a disc of cardboard, thick paper, or tin. Should slag get into the tuyere pipe, it immediately burns a hole in the diaphragm and the blast escapes with a noise that attracts the attention of the furnace keeper. The screws, *c*, can be loosened in a moment so as to insert a new diaphragm. In practice short tapered plugs of soft wood tapped in by a light hammer are often substituted for the discs of cardboard after the first time of burning through.

The ordinary tuyeres employed are flush with the inside of the jackets, but the furnaces at *Aurora* (Ills.) and the whole of those at the various *Broken Hill* plants (Australia) were peculiar in having water-tuyeres which extended into the furnace on both sides, and so narrowed the zone of fusion. At *Aurora*, the side jackets were straight (*i.e.*, had no bosh), and the tuyeres projected 10 to 12 inches into the furnace; at *Broken Hill*, the side jackets had the usual bosh, and the tuyeres projected inwards only 6 to 8 inches. The object in both cases was to decrease the amount of wall accretions and effect at the same time a trifling economy of fuel.

At the *Port Pirie* plant of the B.H. Proprietary Co. ten years ago the furnaces were provided with water-tuyeres of the usual type (shown in Fig. 103), but in the new furnaces these have been replaced by ordinary types of flush tuyeres, retaining, however, the exceptional width which was formerly characteristic of the furnaces in which water-tuyeres were employed.

The old-fashioned tuyere pipe is subject to more or less leakage of air,

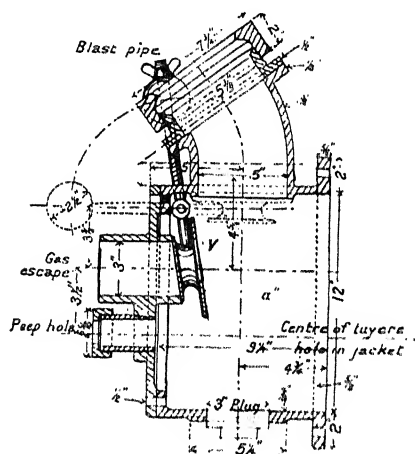


Fig. 119.—Eilers' Tuyere Box.

and several forms of tuyere box have been introduced with the object of making a handy air-tight joint. One device, formerly in common use, is that of *Devereux*, shown in Fig. 106: it consists first of a bored and turned bronze tube fitting into the aperture in the jackets (which must be made specially large if this tuyere is to be used), and inside which an iron sleeve with a skew bore is fitted so that it can be revolved. The blast can, therefore, be turned in any direction (except the most generally useful namely, centrally), which was supposed by its inventor to be a great advantage. The weight of the *Devereux* tuyere renders it clumsy, and in the author's opinion the advantages of being able to alter

the direction of the blast within such limits are at best problematical.

When, from any cause, the blast is stopped, unless the valves connecting the tuyere pipes with the blast pipe are immediately closed, or a suitable safety valve is provided, combustible gases from the furnace travel back into the pipes, mix with the air, and often form an explosive mixture, which, taking fire from the tuyeres, may wreck the blast main and apparatus. Such serious accidents, as described by Austin,\* with their attendant delay and expense, have led to a great deal of ingenuity being devoted to the provision of automatic tuyere valves which shall establish connection with the outer air the moment the blast pressure is taken off. One of the best of these is that of Eilers.† It consists of a simple box fixed against the back of the water-jacket by means of patch bolts, the actual tuyere being simply the aperture in the jacket itself. The construction is evident from Fig. 119, in

\* *E. and M. J.*, Dec. 15, 1894

† Figured in Hofman, *op. cit.*, 1906, p. 233.

which V is a valve with planed face fitting the 3-inch tube in the back of the box air-tight by means of the pressure on its back and rather more than counterbalanced by the iron ball. When the pressure is removed this ball comes into play, and the valve, taking the position shown by the dotted lines, closes the blast pipe and opens the escape. In a line with the tuyere opening is the peep-hole provided with eyepiece as usual, and at the bottom of the box is a 3-inch hole for the removal of slag accidentally coming into the tuyeres, to which can be fixed a fusible plug, if desired. The whole arrangement is compact, though rather cumbersome and expensive, on which grounds it has not come into extended use. Other forms of patent tuyere boxes are described by Hofman.\*

Another form of tuyere box provided with slag-escape, and in which the

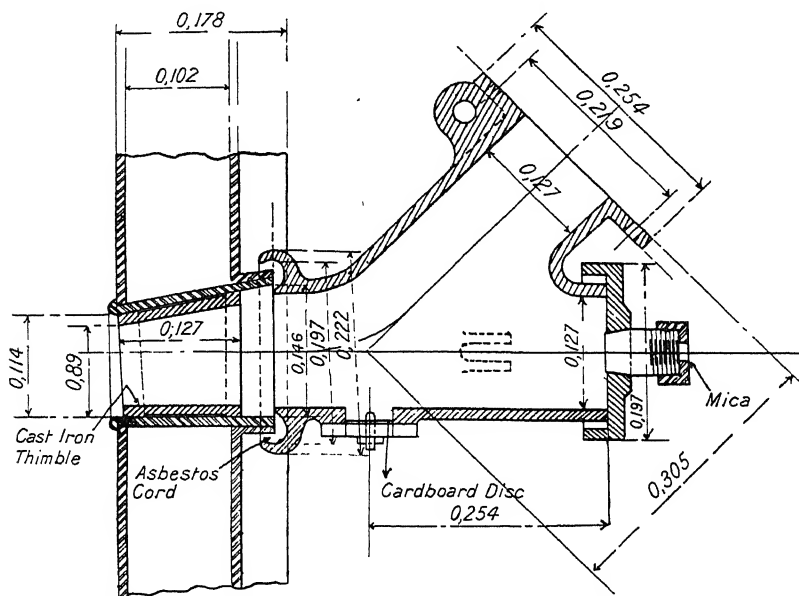


Fig. 120.—Tuyere Box (cast iron). Dimensions in Millimetres.

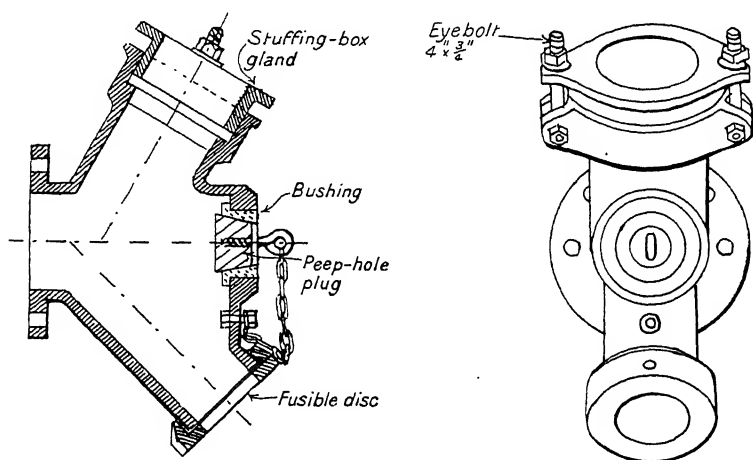
peep-hole is somewhat nearer to the tuyere than in that shown in Fig. 118, is illustrated in Fig. 120.

A very convenient form of tuyere, which has found much favour in Mexico, is the Robinson air-tight non-slugging tuyere shown in Figs. 121 and 122. This has a downward extension 4 inches in diameter, closed by a disc of fusible metal, which melts before the slag actually reaches it, and since the slag by this arrangement can pour out freely, neither the peep-hole nor the air inlet can ever become completely obstructed. The peep-hole takes the form of a plug attached to a chain, and held in place by the pressure when the blast is on. Another advantage of this tuyere is that it is practically air-tight; at Mapimi the saving in blast resulting from the adoption of this tuyere is said to be nearly 20 per cent.

\* *Op. cit.*, pp. 253 to 257.

Separate tuyere valves are always convenient, and do away with the older methods of regulating the blast by tying up the bags or plugging with clay; they are generally placed immediately above the tuyere box, but sometimes, as shown in Figs. 99 and 100, just below the bustle pipe, where, however, they can be readily reached by means of a hooked rod.

**Blast-gate.**—The ordinary form of blast-gate is a simple double-faced disc-valve of wedged-shaped section, like those used for water, but much lighter, and worked by a long handle, pulled in and out by hand instead of being screwed in through a nut, which would be too slow. The weight of the valve is often counterbalanced, in order to enable it to work more easily, and in large sizes the handle is moved by means of a rack and pinion instead of directly. An automatic safety valve opening outwards should be always provided on either the blast main or the bustle pipe, so that the blast may escape whenever, for any reason, it becomes necessary to take blast off the furnace suddenly, without communicating with the blower room. Figs. 123

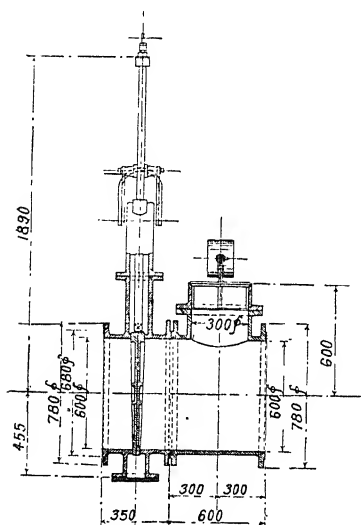


Figs. 121 and 122.—Robinson Air-tight Non-slugging Tuyere.

to 125 show a convenient arrangement of counterbalanced blast-gate and automatic-balanced safety valve. The former is closed from the lower floor by unhooking a light chain, the counterpoise being overweighted; and, as it closes, the safety valve opens simultaneously. If, for any reason, the blower should stop, the blast-gate is immediately closed, which obviates the return of furnace gases with the consequent dangers referred to. The return of furnace gases may also be guarded against by automatic inlet valves upon the bustle pipe, which are kept closed by anything approaching the normal blast pressure, but open downwards by their own weight whenever the pressure runs down owing to the blast-gate being closed.

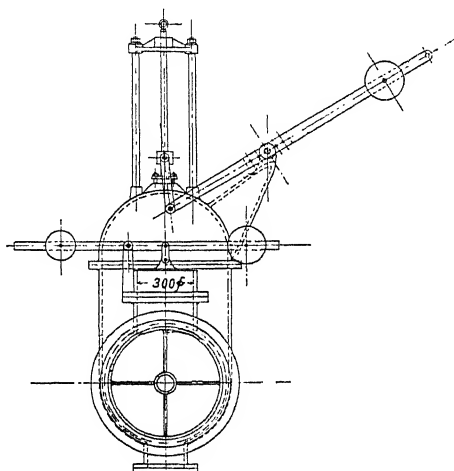
**Blast.**—The blast required in lead smelting is less than for a furnace of equal area at tuyeres engaged in matte smelting; the “hearth efficiency” is lower, the furnace running more slowly. In ordinary matte smelting as much as 300 cubic feet per minute per square foot of hearth area is frequently

supplied, sometimes even more, but in lead smelting from 110 to 220 cubic feet per minute per square foot of area may be allowed, the higher figure referring to easily smelting mixtures and self-fluxing ores with high blast pressures, and the lower to charge mixtures consisting largely of siliceous ores with limestone to flux them, and lower blast pressures. The size of



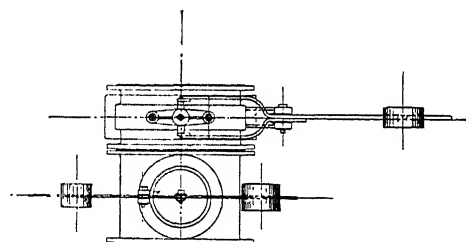
Section

Fig. 123.



Elevation

Fig. 124.



Plan

Scale of Feet

1 2 3 4 5 ft

Fig. 125.

Figs. 123 to 125.—Balanced Blast-gate and Safety Valve.

blower for a given size of furnace will be determined by the above allowance, in view of the nature of the charge which it is expected to smelt; it is well, however, to err slightly on the safe side, since less trouble is likely to be experienced from running blowers a trifle under rather than a trifle over their rated speed.

The following figures from recent practice may be of interest :—

Works.	Dimensions at tuyeres.		Pressure. oz.	Blast supplied.		
	Size	Inches.		Sq. ft. area.	Cubic ft. per min.	
					Total.	Per sq. ft. area.
Port Pirie, . . .	62"	× 212"	91.3	30	9,375	105
Mapimi, . . .	46"	× 162"	51.7	32	6,000	115
Anonymous, . . .	48"	× 200"	66	40	9,500	144
Laurium, . . .	44"	× 156"	47.3	45-50	10,500	220

Attention should be drawn to the fact that, owing to leaky tuyere pipes and valves, the quantity of air entering the furnace is frequently much less than that delivered by the blower, and this is an argument in favour of using air-tight tuyere boxes; reference has been already made to the fact that at Mapimi the Robinson tuyeres are said to save 20 per cent. on the quantity of blast necessary when the old canvas pipe connections to Devereux tuyeres were in use.

In a large works with several blast furnaces there is some economy in running all the furnaces from a common blast main of ample dimensions, regulating the amount delivered to each by blast-gates on the several blast pipes; this arrangement is both simpler and cheaper than giving each furnace its own blower, and part of the blowing power calculated on the above basis is then always in reserve. The blast main should be one-third greater in sectional area than the combined areas of the blower outlets.

Although the common blast main, with branch pipes, is so generally used and so convenient, attention may be called to the fact that it is impossible on this system to regulate the working of each individual furnace to such a nicety, especially when running on different charges, as by providing each furnace with its own blower. The latter system has been advocated in Australia by Greenway (Port Adelaide), and in America by Eulich (Aurora). On this principle the pressure gauge attached to each furnace becomes a delicate barometer, showing accurately the changes in the mechanical or physical condition of the column of charge above the tuyeres, which it never can be when utilised partially for the purpose of regulating the speed of the blowers or the number of individual blowers, delivering into a general blast main, which should be started or stopped. The best plan of all is to provide one blower connected direct to each furnace, but arranging at the same time a general blast main with cross-connections to the furnace branches, so that while ordinarily each furnace is supplied by its own blower it can be supplied alternatively from the general blast main; under this system one spare blower connected to the general main can be switched in so as to allow of each of the others being stopped in turn for necessary adjustments.

**Blast Pressure.**—Not many years ago pressures of  $1\frac{1}{2}$  to 2 lbs. per square inch were regarded as high; in the high furnaces of the most modern practice, however, pressures of from 40 to 50 ozs. are commonly employed. The blast pressure which it is proposed to employ in a new furnace now being designed for the Laurium Company, 64 inches in width between tuyeres, is  $3\frac{1}{2}$  lbs.



Besides the gauges on each blower in the engine-room, each furnace bustle pipe should be tapped for a small nipple, to which a pressure gauge is affixed by means of india-rubber tubing, the pressure being measured by a column of water, glycerine, or mercury: each  $\frac{1}{8}$  inch of mercury or  $1\frac{3}{4}$  inches of water equals a pressure of 1 oz. per square inch.

The power required naturally varies according as the pressure against which the blast is delivered. For each 1,000 cubic feet of free air per minute delivered the power absorbed by the blowers may be reckoned approximately as follows:—

For 24-oz. pressure, . . . . .	about 8 H.P.
32-oz.     ,,     . . . . .	" 11   "
40-oz.     ,,     . . . . .	" 14   "
48-oz.     ,,     . . . . .	" 16   "

**Blowers.**—The blowers employed in lead smelting work are nearly all of the Root pattern with its modifications (Connersville, Cycloidal, Green, &c.). The Baker blowers formerly employed were very inefficient at pressures over 8 to 12 ozs., and are now quite obsolete. The older types of Root's blowers also were much less efficient than those now made, particularly when somewhat worn. Thus, tests made at Rio Tinto \* on a No. 8 Connersville blower showed a "slip" of 35 per cent. at 30 ozs. pressure. These tests, however, were made by the method of throttling the discharge valve to give the pressure, and then determining the speed at which the blower must be revolved to just hold that pressure without any discharge, save that due to slip alone. The method in question always gives high results, inasmuch as the slip is, proportionately to the discharge, very much less at high than at low speeds. A much better means of determining blower efficiency is by the use of the *Pitot* tube.† According to some German experiments‡ on the efficiency of small Root's blowers at high speeds against pressures of 16 inches up to 75 inches of water, the volumetric efficiency when running at 400 revolutions per minute only fell off from 96 to 80 per cent.; so that at the maximum pressures in use in modern lead blast furnaces, say 45 to 50 ozs., the efficiency of the blower may be taken at not under 80 per cent.

The comparatively low efficiency of the Baker and other old-fashioned blowers of the impeller type led Iles§ boldly to discard them altogether in favour of the vertical overhead blowing engines, so familiar in connection with iron blast furnaces. The first cost of these is undoubtedly heavy, but Iles claimed that the efficiency and economy in working more than made up for the heavy interest charges. The same view as to the superior efficiency of the piston blower has been advocated by Vezin and other engineers. The demand for rotary blowers suitable for high pressures has, however, been met by the construction of types of impellers very superior in workmanship (and therefore in efficiency) to the earlier types; and few metallurgists are now inclined to consider the somewhat doubtful advantages offered by piston blowers as sufficient compensation for their much heavier first cost. At the newest American Smelting Works at *Garfield* and *Murray*, near Salt Lake City (Utah) rotary blowers have replaced the piston blowers of the original design, giving

\* *Private Notes*, 1905.

† *E. and M. J.*, Dec. 8, 1906, for figure and description.

‡ See *E. and M. J.*, Sept. 1904, p. 351 (from *Stahl und Eisen*).

§ *Globe Works*, Denver, 1897, *Private Notes*.

every satisfaction; and, at the present time, considering that rotary blowers have also been installed at all the new large copper smelting plants, including Anaconda, Cananea, Douglas, Mt. Lyell, Rio Tinto, &c., where the pressures employed are at least as high as those in use in lead smelting, he would be a bold man who should claim that there is any advantage in piston blowing engines for the moderate pressures employed in lead blast furnace work.

Centrifugal fan blowers, direct driven by steam turbines or electric motors, have been recently introduced by the General Electric Company for pressures of 1 to 4 lbs., and it is claimed that their efficiency varies little between 25 and 125 per cent. of their rated load, but the author is not aware that they have yet come into use at lead smelting works.

At Mt. Lyell, Rio Tinto, and elsewhere very large Parsons' turbine blowers, direct driven by steam turbines, have been installed, with capacities up to 50,000 cubic feet per minute for each unit, delivering air at 60 ozs. pressure to the pyritic smelting furnaces. There is no reason why such blowers should not be applied in lead smelting with great resultant fuel economy.

Fig. 126.

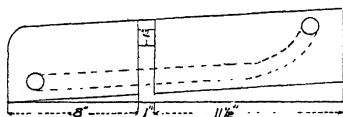


Fig. 128.

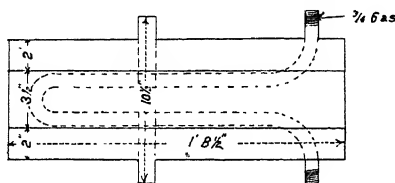
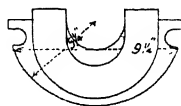


Fig. 127.

Figs. 126 to 128.—Water-cooled Slag Spout (cast-iron).

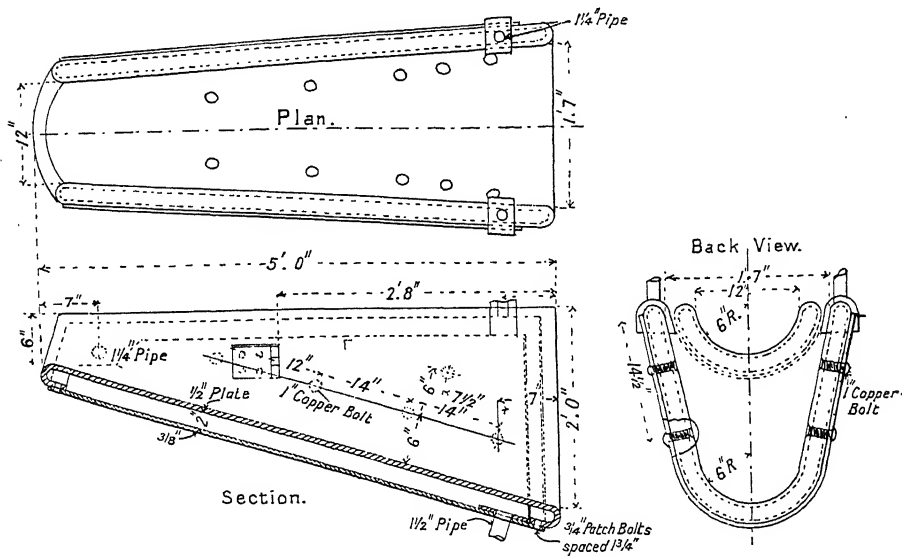
#### HANDLING OF PRODUCTS.

The means whereby the principal product—the work-lead—is removed from the furnace, and its handling, will be described in the chapter on “Products.” The other products of the furnace are slag, matte, and occasionally speiss, which, however, generally accompanies the matte in quantity too small to separate from it. Occasionally, as will be seen, attempts are made to draw off matte and slag from the furnace by means of separate tapholes, but usually they issue together through a single spout, and then separation is effected outside the furnace.

**Slag Spouts.**—The wear and tear on slag spouts is very heavy, the hot matte and speiss in particular which issue with the slag having a corrosive effect upon iron. When small furnaces are used having a capacity of 100 tons per day or so, and the slag is “tapped” at intervals, being shut off by a plug of clay in order to allow its level to rise nearly to that of the tuyeres before tapping again, the spout gets time to cool off between tappings, and ordinary

cast-iron spouts have a life of several months. When, however, the slag stream is allowed to issue from the furnace continuously or nearly so, the flow being controlled either by a succession of small clay plugs, by a heavy iron "dolly" in the taphole, or by "trapping" the blast and allowing the slag to overflow, the wear and tear at a steady red heat becomes too much for plain cast iron, and recourse is had to water-cooling.

*Water-cooled Slag Spouts.*—The simplest form of water-cooled slag spout, when it is not required to trap the blast, is one which retains the form of the ordinary plain cast-iron slag spout, but in which a bent wrought-iron  $\frac{3}{4}$ - or 1-inch gas pipe is placed in the mould at the time of casting, so as to form an integral part of the spout; through this pipe a stream of water is passed. A convenient form is that of the Broken Hill furnaces shown in Figs. 126 to 128.



Figs. 129 to 131.—Water-jacketed Trap Slag Spout (copper).

When there is not too much zinc in the charge, so that continuous flow of slag can be adopted without danger of accumulations of mushy matte on top of the crucible, trap spouts can be employed as in matte smelting. These are naturally more exposed to heat and corrosion than are the plain overflow spouts, and must, therefore, be completely water-jacketed, the water space generally ranging from  $1\frac{1}{2}$  to  $2\frac{1}{2}$  inches in width. Water-jacketed spouts may be either of mild steel or of copper, the latter being much less readily corroded, and, therefore, preferable for continuous duty, especially when much matte or speiss is made. Figs. 129 to 131 show a very good form of water-jacketed trap spout made entirely of sheet copper  $\frac{1}{8}$  inch thick on the inside skin and  $\frac{3}{8}$  inch on the outside; the sheets are turned over so that no seams are exposed

to the hot slag, and the joints are lap-joints fastened by patch-bolts and brazed.

For very heavy wear—i.e., for passing a large tonnage of hot slag—it is advisable to make even overflow spouts completely water-jacketed, and of copper. The long overflow spouts of the Laurium works shown in outline in Fig. 97, and details of which will be given on a later page, are of this character.

**Handling Slag and Matte from the Spouts.**—The oldest arrangement was to draw off matte, slag, and speiss indiscriminately into a conical pot, where they settled out into layers which could be separated easily after solidification. When this is done, below the slag is a cake of matte, and frequently at the extreme point of the cone a button of lead also; while if the charge contain much arsenic, so that speiss is produced, it will be found below the matte and above the button of lead.

**Slag Pots.**—The oldest of all forms of slag pot is that shown standing beside the Castilian furnace in Fig. 74. This form is still commonly employed in German, Spanish, and other Continental localities.

An old American slag pot is that shown in Fig. 132; the wheel is of the compression-spoke type, the spokes being of wrought iron, the rim and hub of cast iron. Lubrication is always a difficulty with slag pots, even when

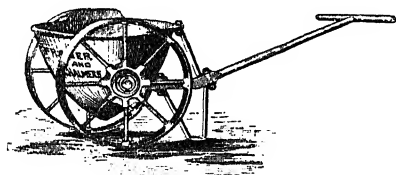


Fig. 132.—Slag Pot (old form).

plumbago is used, and, accordingly, modern small slag pots are nearly always fitted with roller bearings,\* the rollers being of cast steel from 3 to 3½ inches long and from ¾ to 1 inch diameter. The bottoms of the pots usually get worn through in about eighteen months, and to avoid throwing aside the pot as worthless, Terhune† devised a form with

movable bottom, which is illustrated in Fig. 133. Another arrangement for worn-out bottoms, formerly used at the *Parrot Works* (Butte, Mont.),‡ is shown in Fig. 134. It consists of a cast-iron false bottom held in place by a countersunk ¾-inch bolt with large cup-shaped washer, if necessary. It is customary to use new pots for slag, and worn-out ones with the false bottom for matte, producing a somewhat truncated cone which is more easily broken up than the pointed form.

All the above pots have the simple compression-spoke wheel which, according to the author's experience, so soon gets out of order that a plain cast-iron wheel properly proportioned is preferable where a foundry is accessible, since it can be made just as light; it should have a ¾-inch wrought-iron tyre shrunk on to protect the rim.§ Another form of wheel is shown in Fig. 135 with alternate sockets, and, perhaps, as good as any is that described by Keller,|| and shown in Figs. 136 to 138. It consists of a cast-iron hub with wrought-iron spokes and tyre, and, like most modern wheels, runs on steel roller bearings. The hub is tapped to receive the threaded ends of the spokes,

\* *v.* the Terhune pot, Fig. 133.

† Terhune, *Trans. A.I.M.E.*, vol. xv., p. 92.

‡ Keller, *Trans. A.I.M.E.*, vol. xxii., p. 574.

§ Cast-iron wheels are used by the smelters at Port Pirie and Port Adelaide handling Broken Hill ores.

|| Hofman, *op. cit.*, 1906, p. 266.

which are of  $1\frac{1}{4}$  inches round iron, and each of which has a jam-nut to tighten it after the spokes have been adjusted to exactly equal lengths. The wrought-iron tyre is shrunk on and fastened to the spokes by means of countersunk rivets. The splash guards, shown in Fig. 138, prevent the liquid slag from spilling on the hubs.

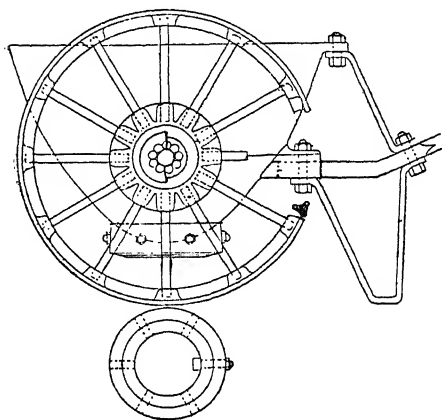


Fig. 133.—Slag Pot (Terhune's).

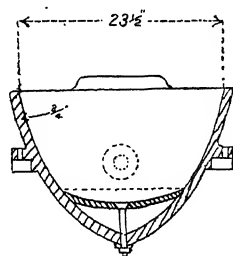


Fig. 134.—Slag Pot used at Parrot Works.

When the slag is left to cool in the pots so as to be broken up and sorted by hand, a large number of pots are required (at least twelve for each furnace smelting 40 tons per day, or more in proportion to the increased capacity) and the wear and tear on them is heavy, while the separation is very incomplete, many small shots of matte being retained by the slag. Most of the shots, however, are contained in the outer portion of the cooled slag, the central core being much poorer, and upon this fact is based the catch-pot system of Devereux, still to some extent in use. Figs. 136 to 138 show one very good form of catch pot with roller bearings and the Keller patent wheels, the axles being fastened by wedges instead of by setscrews; besides the ordinary splash guards, S, it has a projecting lip, L, to take the wear in tipping. At 6 inches above the bottom is a taphole  $1\frac{1}{4}$  inches in diameter, surrounded by a ring and rib reaching the projecting lip in order to protect the pot against injury when the interior slag is tapped by driving a square-pointed  $\frac{3}{4}$ -inch steel bar through the cooled crust. In this way a crust of chilled slag is recovered for resmelting which contains most of the matte shots, while the cake of matte at the bottom of the pot remains undisturbed even if still fluid, which it does not do when an attempt is made to pour out over the rim the interior fluid slag after the crust has solidified.

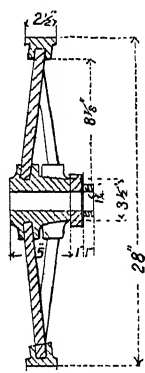


Fig. 135.  
Slag-pot Wheel.

As regards the surface of the dump, some metallurgists recommend laying down tracks of narrow cast-iron plates for the slag pots to run upon, in order

to save labour in pulling the pots considerable distances. The ordinary way, however, to keep the surface of the dump smooth and hard is by levelling a series of squares about 3 feet across with pieces of sheet iron and filling with molten slag, this work being always kept up as the dump advances.

As the size of the furnaces and the distance to the edge of the dump

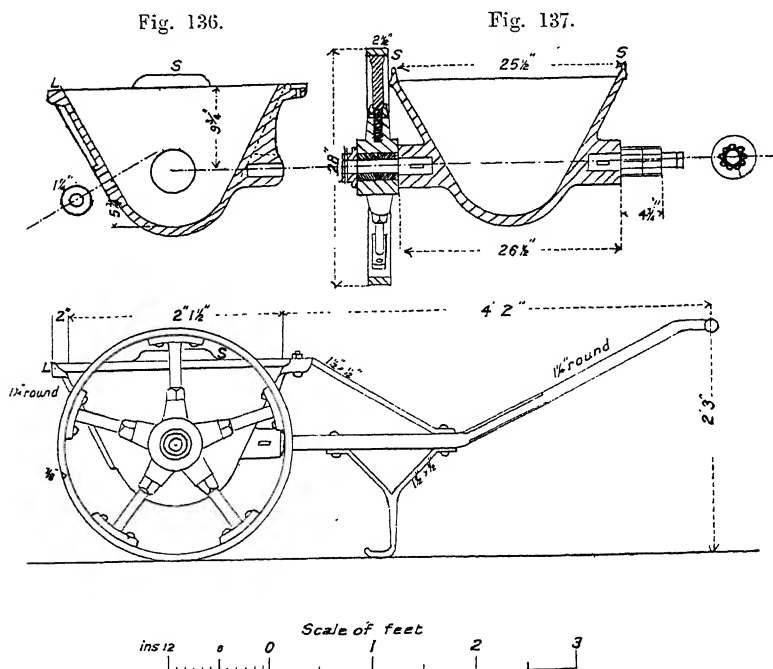


Fig. 138.

Figs. 136 to 138.—Catch Pot.

increase, handling in small pots becomes altogether too expensive, and it becomes necessary to use larger pots running on rails and drawn by horses or by locomotives. These larger pots are of two general patterns, single and double.

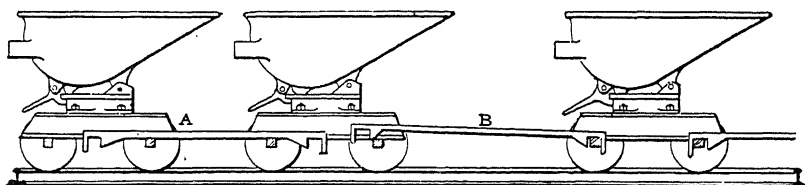


Fig. 139.—Bennetts-Jones Slag Pots.

Intermediate between the hand-pots and those which cannot be handled except by horses, mules, or locomotives, may be described the *Bennetts-Jones* slag pots, shown in Fig. 139. These require tracks (although the wheel-base

Fig. 142.

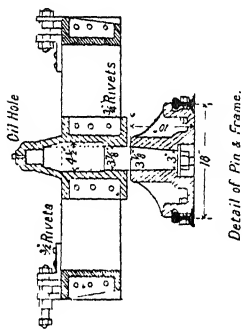
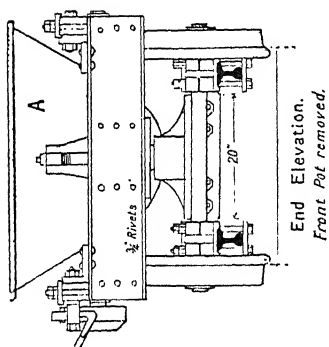


Fig. 143.

Fig. 140.

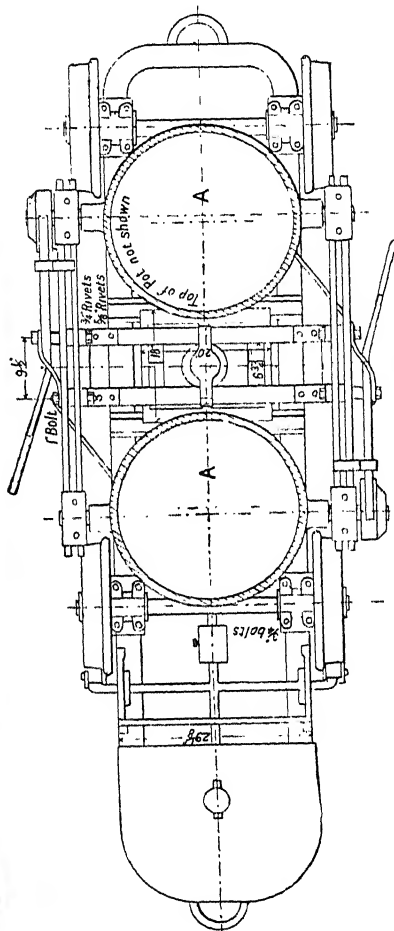
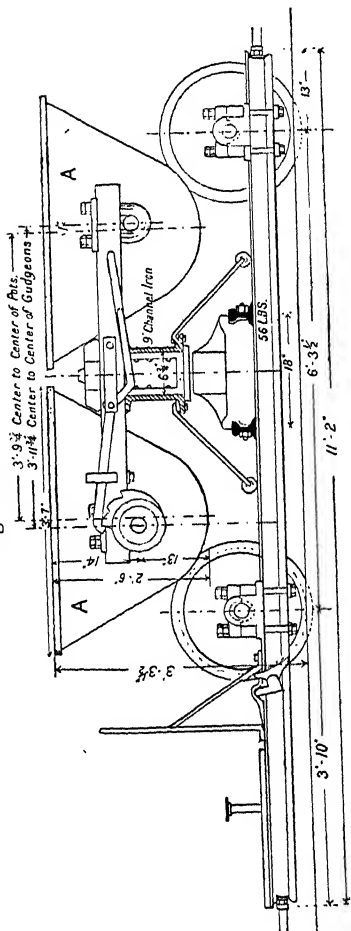


Fig. 141.



Figs. 140 to 143.—Nesmith Double Slag Pot and Truck.

is so short that they will take quite sharp curves), but they may be handled singly if required by one powerful man or two lighter ones; when handled by horses they are made up into trains, as shown in Fig. 139, coupling bars (which, when not in use, rest upon the square axles) dropping into position as the first pot is moved, and so automatically coupling the others. The great advantage of this pot (which holds 1,600 lbs. slag) is that the scoop shape dumps the mass of cooled slag much further from the track than when other patterns are used, and so facilitates keeping the slag-dump in order. The body of the pot is ordinarily made of cast-iron, but the author has found it pays to make them in cast steel, on account of the much longer life so secured.

Of large slag pots requiring horse- or motor-traction, one of those in most

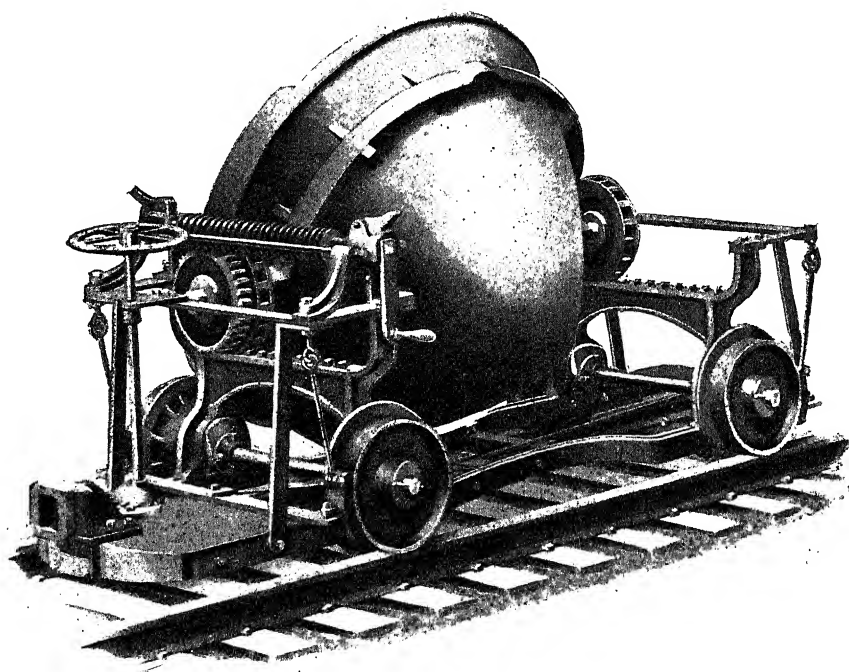


Fig. 144.—Large Slag Pot-truck, with removable Round Pot and fixed Trunnion Ring.

common use in Colorado is the Nesmith double pot-truck shown in Figs. 140 to 143. Each pot has a capacity of  $7\frac{3}{4}$  cubic feet, and holds, when not quite full, about 1,280 lbs. of slag, the total weight of truck, pots, and contents being 5,000 lbs. The pots are filled in the same position in which they travel, and are only slewed round on the central pivot to dump their contents. The slag is delivered at a good distance from the track, which is convenient when molten slag is tipped and shells saved, but the rather long wheel base makes the pots unhandy about a row of furnaces. Larger pots of similar but improved pattern were until recently in use by the Broken Hill Proprietary Company.



At most recently built works large single pots are in use, and are handled by motor or locomotive power. Fig. 144 shows a large round pot in tilting position, the peculiarity of which consists in the fact that the pot itself is removable (and, therefore, renewable when worn out), being attached to the fixed steel trunnion-ring by means of wedges. Hand-driven worm gearing is used for tipping; the capacity of the particular pot figured is 25 cubic feet, or something over 2 tons of slag, but much larger pots are made holding up to as much as 25 tons of slag. A larger elliptical pot with the same kind of tipping gear is shown in Fig. 145, its capacity being 6 tons of slag.

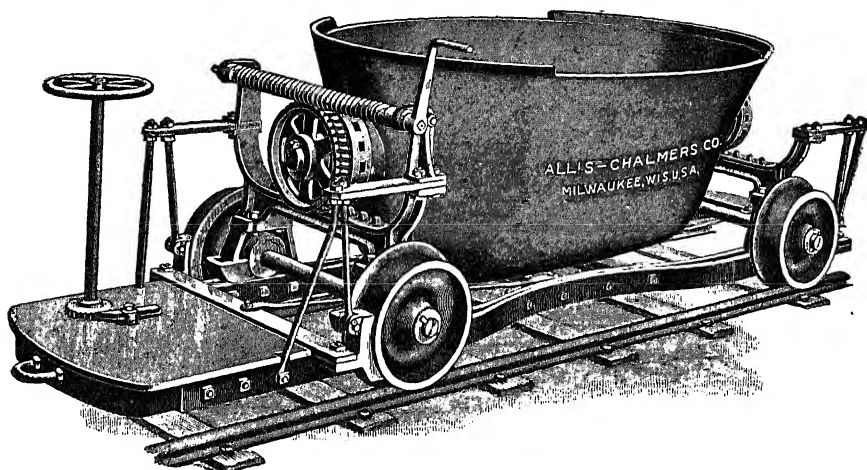


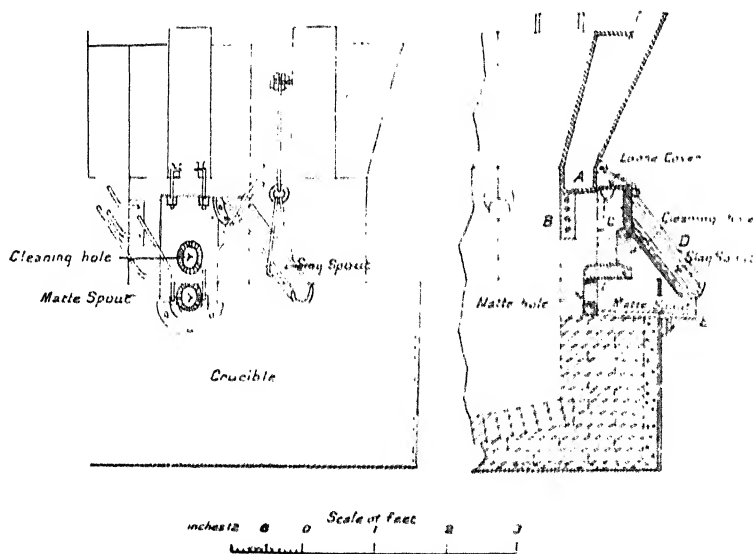
Fig. 145.—Large Elliptical Slag Pot and Truck.

#### SEPARATION OF SLAG FROM MATTE.

**Separation of Slag and Matte.**—The original method of separation of matte from slag, still to be seen in many German, Spanish, and other Continental works, is to receive both together in long conical slag pots in which a separation is effected by density during the cooling process. When solid throughout the cones are inverted, and when cold the tops are knocked off by a gang of labourers, and the matte and foul slag sorted from the clean slag. With the increase in viscosity of the slag caused by higher lime, alumina, and zinc contents, and with the greater care in checking losses caused by keen competition among smelters, it became necessary more than twenty years ago to secure a better separation of slag from matte than is afforded by any mere slag pot; more especially as the percentage of matte produced shows a constant tendency to increase of late years. The methods adopted may be divided into two main classes—A, Those in which separation is sought to be brought about in connection with the furnace itself, the so-called *slag-* and *matte-taps*, and B, those in which the separation takes place quite away from the furnace.

**Slag and Matte Taps.**—As examples of this kind of separating device

may be described the so-called "internal fore-hearths," or matte taps, of Matthewson and Eurich (in use at Pueblo and Aurora), and that used at Broken Hill. The principle in all is the same—namely, the addition to the front of the furnace of a cast-iron water-cooled or firebrick lined box, the bottom of which is open to the slag, while the inner side is formed of a water-cooled "damplate" or "tump." The slag flowing under this diaphragm forms a quiet pool above the level of the tuyeres, thus thoroughly trapping the blast at all times, and overflows through a spout as fast as made into an ordinary settling pot, which catches any shots of matte carried over, and is changed every few hours. From this settling pot, the slag overflows into ordinary slag pots and is considered clean enough to throw away. As soon as matte is seen to come over with the slag from the upper spout a lower taphole is opened, and a potful of matte allowed to run out.



Figs. 146 and 147.—Matthewson Matte Tap.

The "internal forehearth," or matte-tap, of Matthewson is illustrated in Figs. 146 and 147,\* which show plainly the cast-iron damplate, B, with its  $\frac{3}{4}$ -inch gas pipe cast in, as well as the matte and slag spouts. Figs. 148 to 153 show the Broken Hill form of slag tap, in which the end jacket itself forms the back wall of the separating box, the front being formed of a single casting, bolted on and cooled by the usual  $\frac{3}{4}$ -inch gas pipe cast in.

This method of matte separation works most satisfactorily when the slags are fluid and do not chill too rapidly; highly zinciferous slags, however, are not well cleaned, and in order to work satisfactorily the dimensions of the furnace should be such as to pass a minimum of 50 tons slag per twenty-four hours under the damplate.

The matte taps above described form a convenient means of separating

\* *Colo. S.S.M.Q.*, vol. ii., No. 1, p. 40; also *Min. Ind.*, vol. iii., p. 436.

the major part of the matte produced, but they do not succeed in producing matte-free slags, and, therefore, for good work, they must be supplemented by some one of the methods of outside separation next to be described. This fact, coupled with the further fact that the minute globules of matte carried by the slag settle more readily the greater the surface of matte underlying them, has led a majority of metallurgists nowadays to discard altogether interior separation by means of matte taps, and to rely upon exterior separation exclusively. The following methods of outside separation may be described:—

1. **The Overflow Pot.**—This is usually more or less of the slag-pot form

Fig. 148.

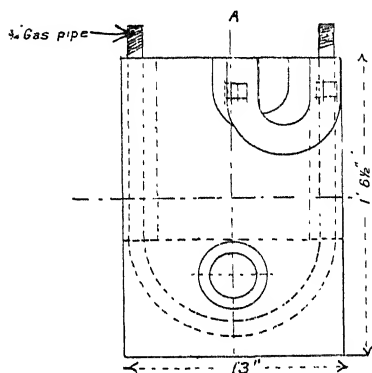


Fig. 149.

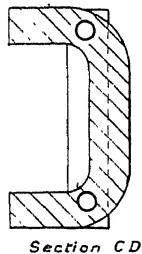
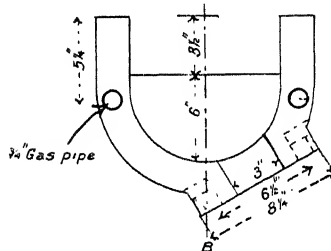
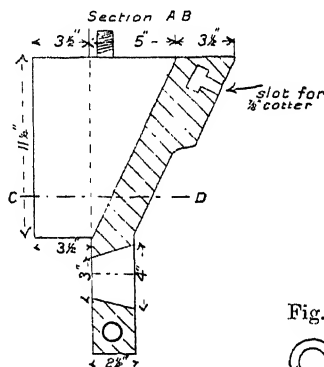


Fig. 152.

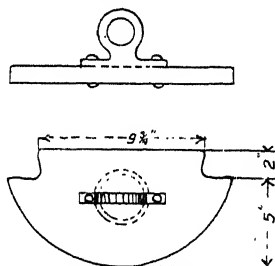


Fig. 153.—Cover.

Fig. 150.

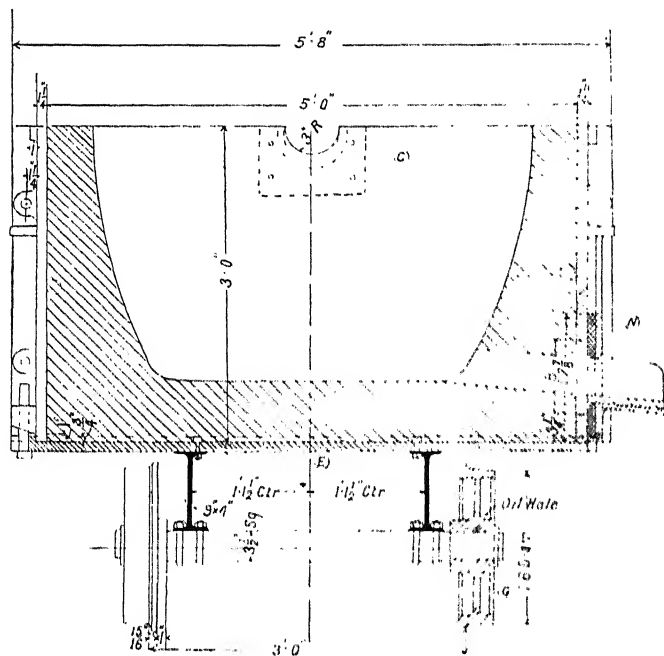
Fig. 151.

Figs. 148 to 153.—Broken Hill Matte Tap.

but much larger, and provided with a cast-iron spout through which the slag can escape after depositing the greater part of its matte. Sometimes a cast-iron cover is provided, but generally a simple crust of slag is allowed to form over the top to prevent rapid chilling of that below. The greater portion of the matte collects in the bottom of the pot, whence it may be tapped through a ring-protected aperture; or, if in insufficient quantity, by allowing the overflow pot to cool and breaking up its contents.

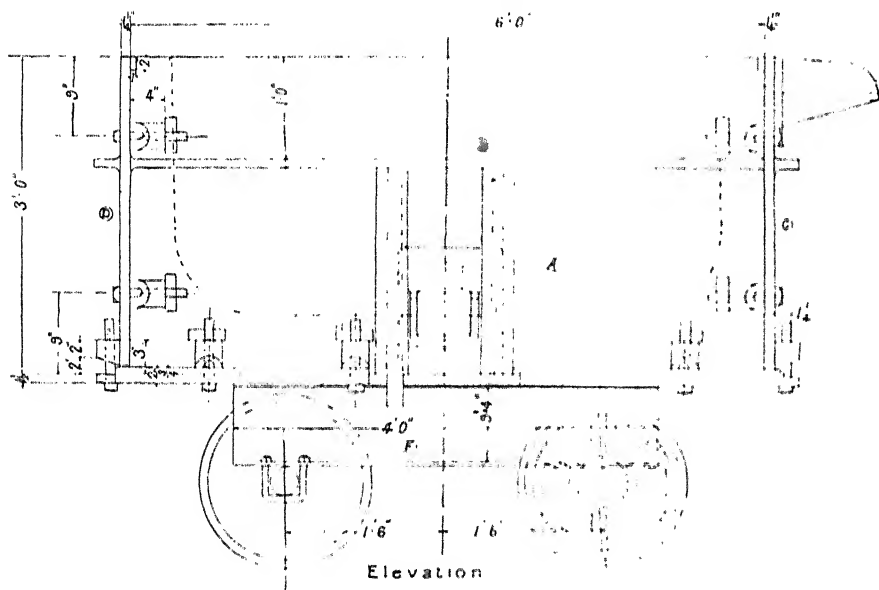
2. **Large Crane-handled Settling Pots.**—The Livingstone system described by Austin \* as in use at the Grant Works (Denver) was figured and

\* *E. and M. J.*, Nov. 23, 1895.



Section.

Fig. 154. —Forehearth (cast iron).



Elevation

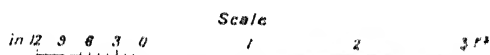
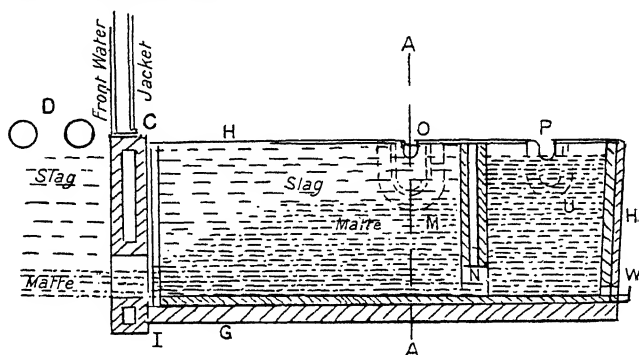


Fig. 155. Forehearth (cast iron).

described in the first edition of this work, and was seen in use by the author so late as 1902, but is now obsolete, and the works where the method flourished are now dismantled.



Longitudinal Section.

Fig. 156.

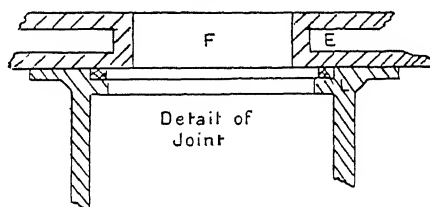
Detail of  
Joint

Fig. 157.

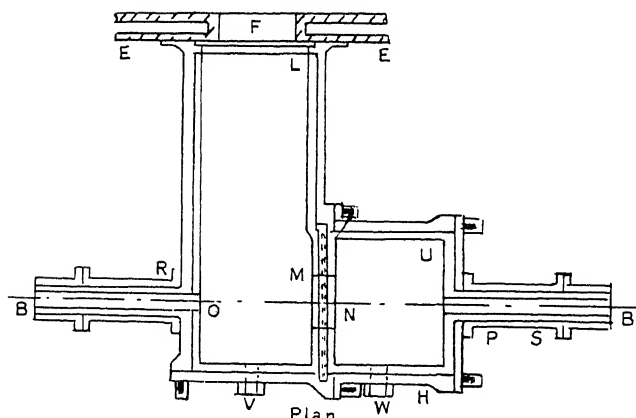


Fig. 158.

Figs. 156 to 158.—Matte Overflow Forehearth.

3. The Forehearth or "Settler."—This appliance, consisting in the main of a cast-iron box on wheels, is in use at a large number of modern works in various sizes, ranging up to 300 cubic feet capacity.

Figs. 154 and 155 show a rectangular forehearth of about 60 cubic feet capacity, built of ribbed cast-iron plates with a brick lining, which takes to pieces, and is very convenient for small furnaces (say up to 100 or 120 tons charge per twenty-four hours).

Another form of forehearth which to the advantages of exterior separation adds that of separate overflow of clean matte, and complete protection of all the small globules of matte from oxidation, whereby efficient settling and coalescence are promoted, is the so-called Oxford forehearth, one form of which is that of the *Hall Mg. and Sm. Co.* (Nelson, B.C.).\* This forehearth, illustrated in Figs. 156, 157, and 158, consists of an L-shaped cast-iron box about 30 inches long, either lined with tiles or water-jacketed, according to the conditions, of a depth equal to the depth of molten material in the crucible of the furnace, provided with a water-jacketed partition reaching to within a few inches of the bottom, and with two spouts on opposite sides of this division, one for the matte and the other for the slag. In the Figs., E is the furnace

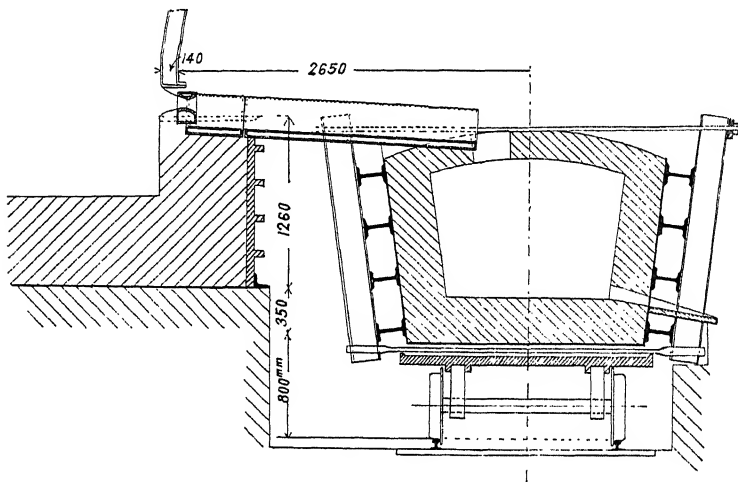


Fig. 159.—Section through Jacket and Forehearth.

tap-jacket, and F the opening in same; L is a groove in the cast-iron plates forming the forehearth tamped with clay to make a joint, M is the water-jacketed division, and N the opening in the bottom of same through which matte alone can pass from the larger division into the smaller one, O and P are the slag- and matte-spouts respectively, V is an emergency taphole for tapping out any lead which may come over with the matte, or for emptying the forehearth and the furnace when necessary. The whole apparatus rests upon wheels which run on two rails placed at right angles to the furnace; according to the inventor it will run for two or three weeks without attention, and, unlike any other forehearth or device for automatically separating matte from slag, it will give good results with as little as 8 tons of matte and 50 tons of slag daily, even when the slag produced is high in iron and in zinc and low in lime.

\* Harris, *E. and M. J.*, Jan. 27, 1906, p. 178.

To make a joint at starting when the forehearth is jacked up into close contact with the furnace, fireclay is tamped into the groove, L, but any leak of slag quickly chills in the crack, and the joint thereafter gives no trouble. The height of the various overflows depends upon the height of the charge column and the blast pressure carried in the furnace, but the amount of overflow at any given moment is to a considerable extent under the control of the furnaceman.

By a little attention the overflow of matte from this forehearth can be made nearly continuous, or by the use of a clay dam single potfuls of matte can be drawn off at any desired short interval.

Figs. 159 to 162 \* show the much larger rectangular forehearth in use at Laurium, the casing of which measures 10 feet  $\times$  6 feet 6 inches  $\times$  4 feet 6 inches in depth, and holds inside its lining of chrome-magnesia brick about 200 cubic feet or over 20 tons of slag and matte. Figs. 159 and 160 show the general arrangement of the forehearth with regard to the furnace; the

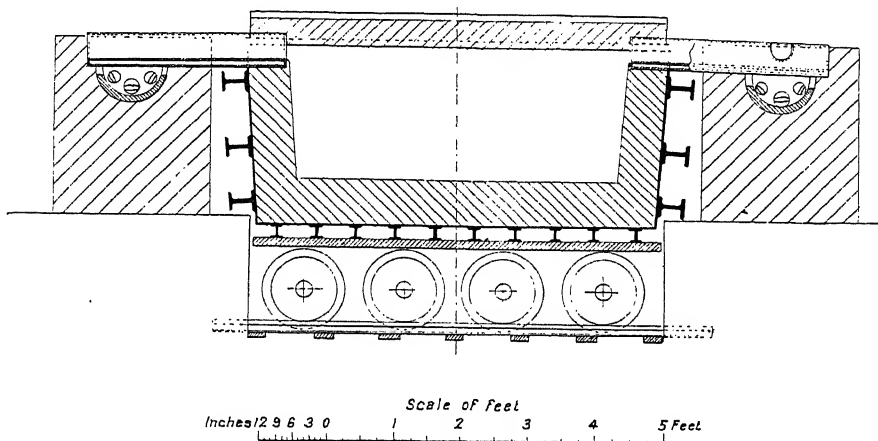


Fig. 160.—Longitudinal Section through Forehearth.

tapping jacket and water-cooled spout have been already described and figured. Figs. 161 and 162 show details of the bogie truck, &c. The casing of this forehearth is of  $\frac{3}{8}$ -inch mild steel, resting upon 4-inch I-beams to get the advantage of air-cooling, these in turn being supported upon a heavy cast-iron bed plate.

The newest furnace at Laurium is to have a still larger fixed forehearth, circular, and 12 feet in diameter inside the refractory lining, like those employed in connection with copper blast furnaces. The arrangements at Laurium for controlling the flow of molten products into the forehearth above figured are novel.† The stream of slag, matte, and speiss issues from the furnace through the phosphor-bronze tapping jacket, detail of which has been shown in Figs. 113 to 115, into the water-jacketed spout, 7 feet 3 inches long, shown in section in Fig. 159. This spout replaces others of cast iron which,

\* Private communication, L. Guillaume, 1908.

† See brief description by the Author in the *E. and M. J.* of May 1, 1909, p. 882.

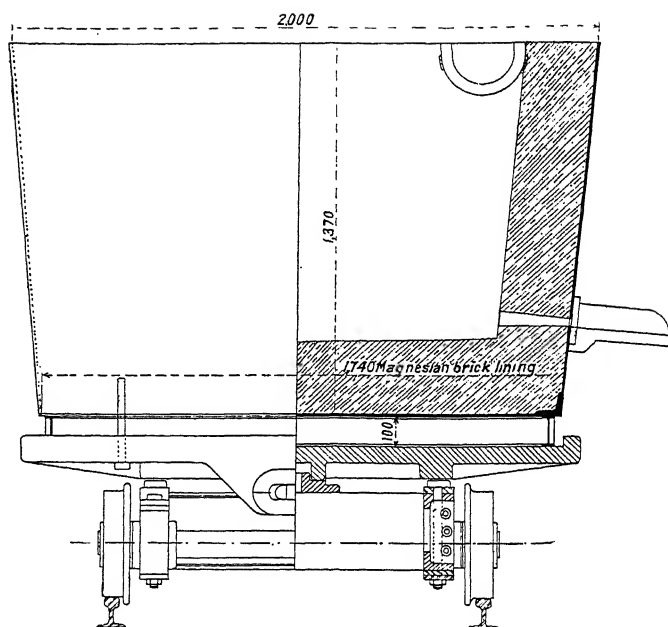


Fig. 161.—End Elevation, Half-section Rectangular Forehearth (Laurium).

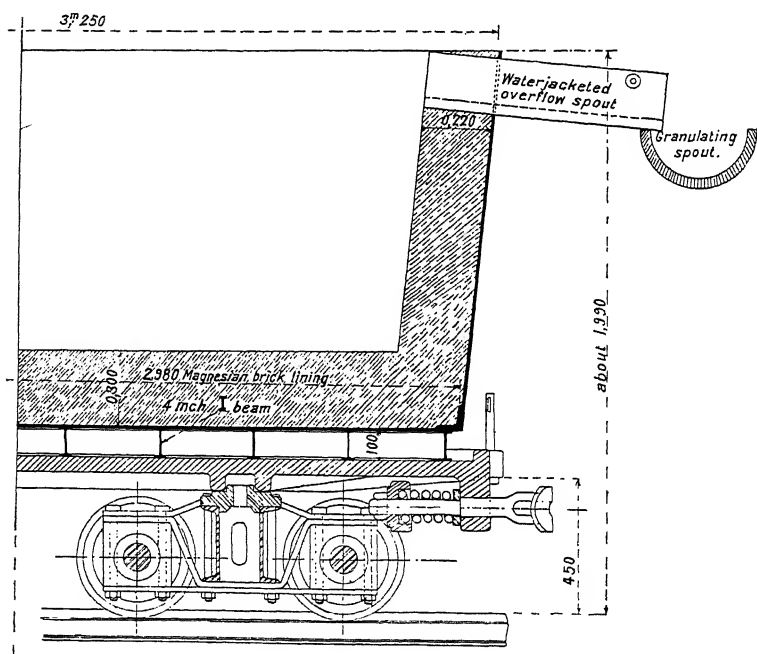


Fig. 162.



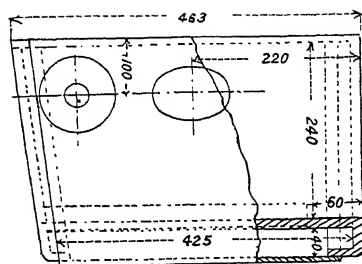


Fig. 163.—Longitudinal Section, first section Spout. (Dimensions in millimetres.)

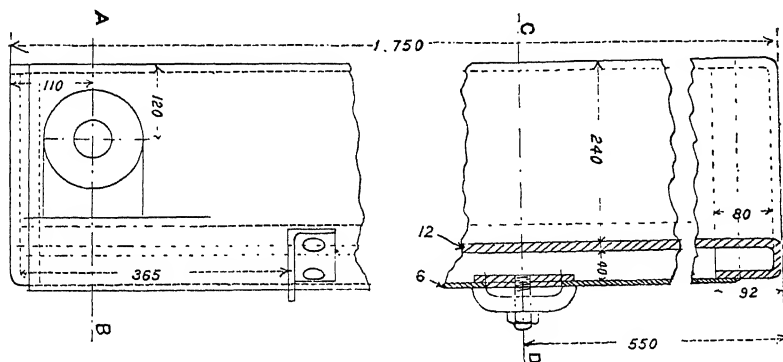


Fig. 164.—Longitudinal Section, second section Spout.

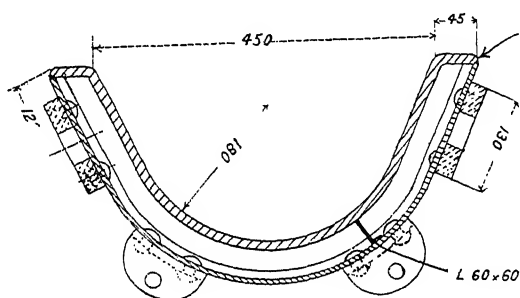


Fig. 165.—Section A B.

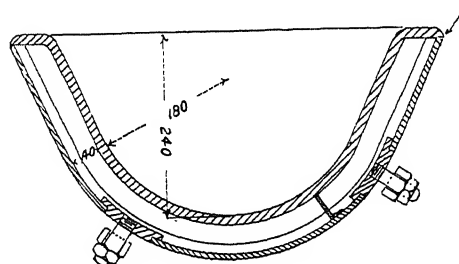


Fig. 166.—Section C D.

whether used bare or lined with magnesia brick, were rapidly cut out by the hot speiss; details of its construction are shown in Figs. 163 to 166, from which it will be seen that it consists of two sections with independent water-supply, each formed wholly of copper plate,  $\frac{1}{2}$  inch thick on the inside and  $\frac{1}{4}$  inch thick on the outside, with the seams all brazed, and no seam being exposed to heat; each section has two hand holes for cleaning, and a baffle along the bottom forces the water to circulate right out to the end. Fig. 167 is an elevation of the tapping jacket and spout.

Instead of adjusting the flow of slag from the breast by means of iron dollies or an attendant with clay balls, the control has been rendered nearly automatic, and the flow perfectly regular by means of the water-cooled plug shown in longitudinal section in Fig. 168, and in transverse section in Figs. 169 and 170. The plug consists of a length of weldless solid-drawn steel tube,  $1\frac{1}{2}$  inches internal diameter by  $\frac{3}{16}$  inch thick, with a hollow spherical point of cast copper,  $\frac{5}{16}$  inch thick at its thinnest part, screwed on to its end. Inside the steel tube is fixed axially a thin tube of sheet brass,  $\frac{3}{4}$  inch internal diameter, with its point flattened horizontally, and extending nearly out to

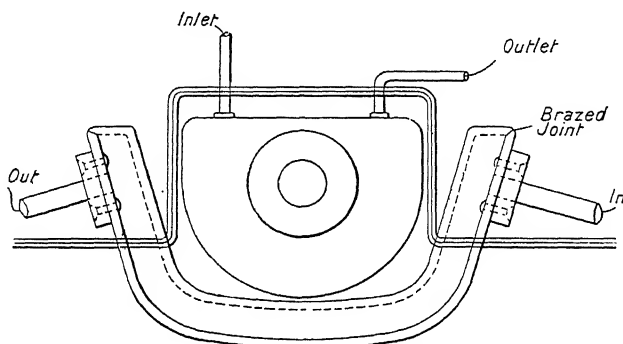


Fig. 167.—Front View of Tap-jacket and Spout.

the inner surface of the globular copper point; by means of an appropriately shaped cast-iron head and connections of india-rubber tubing a stream of water is made to circulate inwards through the brass and outwards through the steel tubing. The whole arrangement is held by means of glands in the centre line of the tapping jacket, in such a way that by a gentle tap with a hammer it can be either pushed in or drawn out, thus restricting or enlarging the area of the annular ring between the rounded point of the plug and the tapered hole through which the molten products issue.

**Perfection of Separation in Forehearths.**—With low-grade lead bullion, carrying not over 0.2 to 0.35 per cent. silver, or say 60 to 90 ozs. silver per ton at the outside, a single separation, as carried out in these large forehearths, is amply sufficient to secure slags low enough in silver to throw away. With an ore charge richer in the precious metals, however, yielding high-grade lead bullion, the slags, after passing through one such forehearth, still carry with them in suspension minute globules of matte, sufficient to raise their tenor in silver to 1 oz. per ton or over; this is more particularly the case when the viscosity of the slag is increased by high alumina contents,

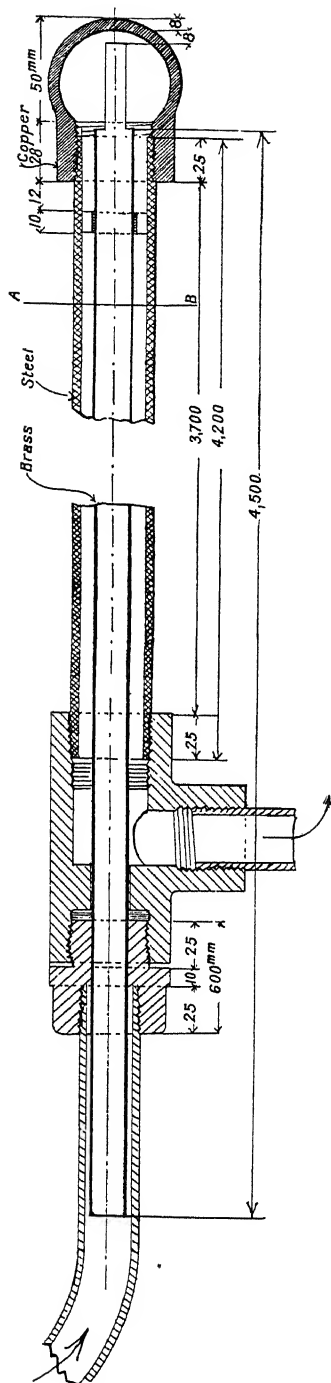
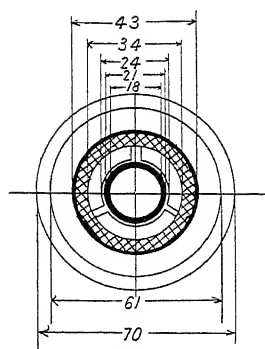
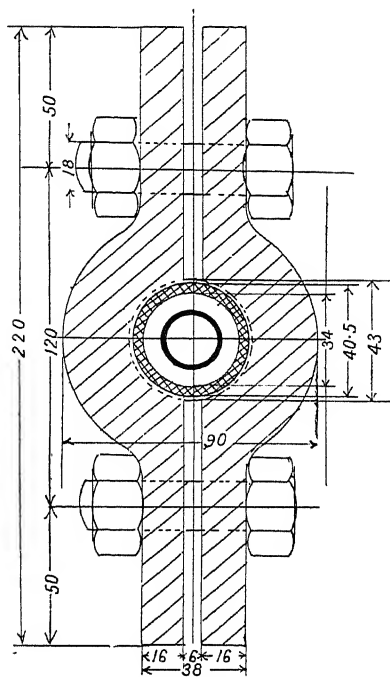


Fig. 168.



Section.A.A

Fig. 169.



Section Through Gland

Fig. 170.

and still more so when much zinc sulphide is present in the charge, the evil effects of the latter substance being felt not only in increasing the density and viscosity of the slag, but in forming a large quantity of comparatively light matte, which separates badly from the slag.

Since nearly all custom smelting plants of any size turn out a work-lead, the precious metal contents of which are worth much more than its lead contents, common figures for American and Mexican plants being from 200 to 300 ozs. of silver, besides from one to several ounces of gold, and since such high-grade work-lead is always accompanied by a matte of correspondingly high grade, the separation of this matte as thoroughly and completely as possible becomes a matter of the greatest economical importance. Nearly all modern works, therefore, have adopted one or other of the two following systems, viz. :—

1. A double separation conducted first in a rectangular forehearth of large dimensions in front of the furnace, in which most of the matte settles out, to be tapped at intervals into pots, and subsequently by settling in large slag pots, the shells of which, containing most of the remaining small globules of matte, are saved for resmelting.

2. Single separation by means of the reverberatory slag separators, first introduced by Rhodes at Leadville, and subsequently adopted at the Globe Works (Denver), at Monterey (No. 3 plant), and at the new works at El Paso and Salt Lake City.

The double separation in forehearths and pots is well exemplified at *San Luis Potosi*.<sup>\*</sup> Slag and matte issue together into long forehearths of cast-iron plates with wrought-iron hooks cast in for lifting off, and blocks or lugs cast upon the bottom plate so as to hold the sides in position, and forming a box 6 feet  $\times$  3 feet  $\times$  30 inches at top with inclined sides flaring downwards. The high wheels are protected against droppings of slag by means of cast-iron covers screwed on to the base-plate. Matte is tapped at intervals from the forehearth into shallow bowls, which rest in a circular hole in the top plate of a low truck running on a track uniform with that of the slag pots and forehearths. The slag from the forehearth, in which most of the matte has separated, overflows into 33 cubic feet (3 tons) elliptical slag pots, which are provided with a taphole and spout near the bottom. These are pulled out to the dump by a mule, and, after standing for a period of from ten to twenty minutes (varying according to the temperature and degree of fluidity of the slag made at any given time), the slag is tapped out and the pot is run back to a dumping platform, where the shell, varying from 1 to 2 inches in thickness at the sides up to perhaps 6 inches on the bottom, is dumped, broken up, and shovelled into iron cars, which are pulled up an incline to a hopper above the feed floor by means of a wire rope and friction drum. The pot-shells contain on an average from 7 to 10 ozs. of silver per ton, and from  $1\frac{1}{2}$  to  $3\frac{1}{2}$  per cent. of lead, contained in the small shots of matte and bullion which have gradually coalesced and settled out, the average contents of each shell being enough to give it a nett value for resmelting of about \$1.00 U.S. Cy., while the cost of handling is trivial. The matte bowls are taken to a matte yard, where there is a fixed derrick, and before the matte sets a hook is inserted in each. When set the derrick lifts out the blocks of matte, which are sledged, and then sprinkled in piles to weather them so as to facilitate the crushing.

<sup>\*</sup> *Private Notes*, 1902.

The double separation in forehearth is also carried out at, amongst other places, *Murray* and *Bingham*. At the latter plant the overflow slag from the forehearth is received in large Nesmith single pots provided with Devereux tapholes; these are tapped on the dump, and all the shells are saved for resmelting. One advantage of the tapping is that it makes sure of saving the matte should there be, owing to carelessness in not tapping the forehearth at the proper time, a considerable quantity in one or more slag pots; whereas when the pots are dumped to save the shells there is always the probability that any quantity of matte would remain melted owing to its low freezing point in the bottom of the pot, and be thrown away with the waste slag.

4. **The Reverberatory Settling Furnace.**—Messrs. Rhodes, late of the *Arkansas Valley Works* (Leadville), and Iles, of the *Globe Works* (Denver), have worked on the principle of an external source of heat for keeping the slag perfectly fluid, so as to give shots of matte a better chance of settling out. In both plans the reverberatory (which should preferably in a large works be duplicated to allow of cleaning out and repairing from time to time) has a separate fireplace, is water jacketed, is situated at a distance from the furnaces, and has separate tapholes for slag and matte, the latter being low down on one side near the firebox, while the former is either on the opposite side (Rhodes) or at the far end (Iles).

In the Rhodes separator (*Arkansas Valley*) the slag from the furnaces is brought in hand pots and poured in through a lip at the end, the shells (10 per cent.) being returned for resmelting. The capacity of the furnace is 100 tons of slag, which comes direct from the slag tap of the furnaces, it being found that the ore-matte is required in order to collect and settle that carried by the slag in suspension. The slag is tapped every half hour into a train of Nesmith double-pot trucks drawn by a locomotive; the matte is tapped into hand pots and granulated for roasting.

At the *Globe Works*,\* where the Iles separator is used, the slag from the furnaces is tapped at intervals into a large and long settling box of 120 cubic feet capacity with a side taphole for matte, which at these works is 8 to 12 per cent. of the charge. The slag falls into double-pot trucks, which are pulled uphill away from the furnaces by a mule, drawn up by means of a steam winch and wire rope to a platform of I-girders over the separating reverberatory, which has in its roof a square cast-iron hopper to guide the charge of molten slag. One of the blast furnaces is unprovided with settling box, and from it the whole of the slag and matte are tapped together and brought to the reverberatory for separation, the matte so brought amounting to 3 tons per day out of a total of 8 tons yielded by the reverberatory. The slag-matte is found to be 4 or 5 per cent. richer in copper and proportionately richer in lead and silver than the ore-matte.

The following points should be noted in connection with slag-settling reverberatories:—†

1. They should be built of good firebrick with extra thick walls completely encased in 1½-inch plates and strongly bound; the hearth should be built inside a complete wrought-iron pan to keep leakage as low as possible; as it is impossible to make this completely lead tight, Iles recommends below

\* *Private Notes*, 1897.

† Iles, *E. and M. J.*, Dec. 9, 1899, p. 695.

it at least 6 feet of well tamped clay in layers, as being more impervious than the ordinary brick foundation, followed by about 1 foot of red brick.

2. Although it is unnecessary to water-cool the side walls of the furnace, the bridge should be cooled either by a water-jacket (Rhodes) or by several independent 3-inch pipes running through it from end to end, each having its own water supply.

3. Pot-shells should never be put in, and the temperature should be kept as even as possible, firing small quantities of coal at frequent intervals.

4. The tapping block for matte should be a heavy block with water pipe cast in, and should be kept as close in to the matte bath as possible to make easy tapping, pine plugs being used if necessary. The matte should never be tapped out dry, a pool of 5 to 8 tons being always left in the bottom.

The capacity of the reverberatory settler should be so proportioned to the slag output of the blast furnaces as to allow an average of from forty-five minutes to one hour for settling. As the slag cools but little in transit from the blast furnaces, the consumption of fuel is small, not over about 1 per cent.

At the *Arkansas Valley Works* slags entering the separator averaged in 1896 0·9 per cent. Pb and 0·8 oz. Ag per ton, those leaving it averaged 0·6 per cent. Pb and 0·5 oz. Ag per ton, showing an average saving of 0·3 per cent. Pb and 0·3 oz. Ag per ton.

At the *Globe Works* in 1898 \* the cost of handling 237,892 tons of slag in two separators was per ton—Labour, 4.6c., coal 2.65c., repairs (labour and materials) 2.26c.; total, 9.51c. per ton (of 2,000 lbs.) slag passed through.

The saving effected was as follows :—

	Assays.		Contents.	
	Percent. Pb.	Oz. Ag.	Lbs. Pb.	Oz. Ag.
Slag entering settler, .	0·67	0·88	3,173,409	208,816
Slag leaving „ .	0·37	0·55	1,753,937	130,191
Saving, . . .	0·30	0·33	1,419,472	78,725

The value of the saving effected was—

	Dollars.	Per ton. Cents.
Gross saving, . . . . .	86,738	36·5
Less cost, . . . . .	22,625	9·5
	64,113	27

Even assuming some tendency for the assays to favour the separation, owing to the samples of “slag entering” carrying occasional globules of matte, the resulting saving is a remarkably good showing.

One of the most complete slag settling arrangements known to the author is that at the *No. 3 Smelter, Monterey†* (Mex.). At these works the whole of

\* *Iles, loc. cit.*

† *Private Notes, 1902.*

the slag and matte from each furnace is tapped at intervals into large 6-ton round slag-pots standing on tracks of 3 feet 6 inches gauge, which, starting out at right angles to the furnace building, all come together at the separator. When one of the 6-ton pots is full a small locomotive drags it uphill to the separating furnace, while another is braked downhill to take its place. The molten slag is poured into the furnace through a lip at the end opposite the grate, and underneath the flue. The shells of the 6-ton pots when too thick are turned out, broken up, and wheeled into the boot of a bucket-elevator, which raises the slag to a hopper above the feed floor. One furnace will easily take over 1,000 tons of slag per day, which is the production of the whole plant, so that having two furnaces in use it is easy to make the slight repairs to lining which are required about every fifteen to twenty days.

The slag-tap is situated on one side of the separator furnace near the bridge end, and consists of a heavy (4 inches thick) cast-iron plate, not water-cooled, and with a 4-inch hole in it. The cleaned slag is tapped almost continuously from it into trains of four elliptical slag pots, each holding 4 tons, and handled by a small locomotive. The matte- and lead-taps are situated one below the other on the opposite side of the furnace to the slag-tap and near the bridge. Matte is tapped about every three or four hours from a similar tapping-block into a large and deep round pot, which holds 7 or 8 tons when full. It is pulled out by a locomotive, and taken to a dumping track where, by means of screw gear, its contents are poured from a lip into 10 or 11 hand pots of about 700 to 800 kilos. capacity, which are handled on an iron-plate floor by three men, and dumped to cool, after which they are broken up and taken to the crusher. The average production of matte from ten furnaces is 50 to 60 tons per day. Lead is tapped once a day from a taphole below that used for matte, and is brought with the matte to the dumping ground and poured into the hand pots. A hook is inserted into each of these, the cake of matte lifted off when set, and the lead is ladled into moulds. The coal consumption is 8 to 10 tons per day for each furnace, or about 1 per cent. on the weight of slag handled, as at the *Globe* works.

At *Mapimi* (Mex.), owing to want of fall in connection with reverberatory settlers, the new furnaces are furnished with a novel arrangement for handling the slag.\* Instead of a forehearth, each furnace delivers slag and matte together into a large 5-ton pot resting by means of trunnions on the lower ends of two short, heavy rails, forming an inclined track, which terminates in a pit below the slag spout. A long bail is attached to the pot, and when full the hook of an electric travelling crane is hooked into the ring on the bail, and the pot is pulled up the incline, swung clear, and carried to the short charging spout of the 80-ton reverberatory settling furnace. Here, by means of another chain hooked into a ring near the bottom of the pot, the contents are poured into the furnace, the pot is raised, taken back, and skidded down in front of the slag spout again.

Arrangements for reverberatory matte separation more or less similar to those described are in use at most of the newer plants, including those at El Paso, Salt Lake City, &c.

One of the great difficulties in the way of effecting complete separation of matte from slag is the presence of zinc sulphide, which, as has been already noted, tends to separate and form a mushy layer between the slag and the

\* *E. and M. J.*, vol. lxxxvi., Aug. 22, 1908, p. 374.

proper matte. The tendency to form this separate layer when the proportion of zinc in the charge is considerable is more noticeable when the lime in the slag is high and the iron low, partly, no doubt, because the specific gravity of the slag is then low, giving the zinc sulphide a better chance to settle out from it. It is often necessary, when working on zinciferous charges, to provide an extra taphole intermediate in level between those of the matte and slag, through which this mushy zinc matte may be worked out after tapping the slag down low. When old tin cans or other scrap iron can be obtained, part of the zinc sulphide can be decomposed by adding them to the forehearth or reverberatory settler, the iron taking up the sulphur to form matte, while metallic zinc is vaporised and burnt.

An additional precipitation of part of the metallic contents of the slag carried in suspension or solution can be obtained by feeding together with it into the forehearth or reverberatory settling furnace a small proportion of finely divided metallic iron in the form of turnings, borings, or thin sheet scrap, or clean iron pyrites. Both act in the same two ways—namely, by precipitating copper, lead, and the precious metals from their compounds in solution, and by collecting the small suspended globules. The use of finely-divided iron scrap in forehearths for the above purpose was first introduced by the author in 1890,\* and his experiments at a somewhat later date proved that finely-divided clean pyrites was even more efficacious as a precipitant and collector, the only drawback to its use being the very low grade of the matte produced, the flux value of the iron in which, however, apart from the metallic contents recovered, will often pay for roasting and resmelting. At the works where the author's experiments were made, the practice never became standard, because clean barren pyrites was not available in quantity. Under suitable local conditions, however, such precipitation of the metallic contents of slags by barren pyrites may become an important feature in the local treatment.

**Slag Granulation.**—This method of disposing of slag, although not so commonly employed in lead smelting as in matte smelting plants, may be utilised to great advantage when water is plentiful, the fall great, and other conditions suitable. In order to prevent the formation of little balls or bombs of slag, which sometimes explode, it is important that the stream of slag should not fall into the water from a height, but should overflow from the spout quietly into a stream of water, preferably issuing at a considerable velocity. If the granulated slag has to be conveyed any considerable distance, it is essential that heavy iron troughs (preferably of cast iron) should be employed, since the stream of slag granules soon cuts through ordinary sheet iron. A slope of  $\frac{1}{2}$  inch or even  $\frac{1}{4}$  inch to the foot is amply sufficient to carry the slag any distance. At the *Hanauer Works* (Utah),† the trough was 150 feet long, 2 feet wide, and 6 inches deep inside the cast-iron bottom, which was 1 inch thick. The pitch of the trough was only  $\frac{1}{4}$  inch to the foot, and with a 13-foot head, a small supply of water in addition to that from the jackets was sufficient to carry the slag from four furnaces to an elevator pit, whence a belt elevator with sheet-iron buckets raised it into a storage bin above the railroad track.

Even without the help of extraneous water, it is possible to granulate slag

\* *v. Proc. Inst. Civ. Eng.*, vol. cxii., pp. 151 and 152.

† *Private Notes*, 1897; also *Terhune, S.Q.M.*, vol. xv., p. 108.



very satisfactorily by means of the jacket water only, the condition essential to success being that the stream of slag shall be delivered upon that of water already flowing in the same direction. This end is attained in a simple manner by the M'Arthur trough, in use at *Sudbury*, and shown in Figs. 171 to 173. Fig. 171 is a side view of the trough in front of the forehearth, which is mounted on wheels so as to be readily removable when changing forehearths. The trough is in two parts, and details of its upper section, which does the granulation, are given in Figs. 172 and 173. The granulating water, which is only the overflow jacket water, enters the lower part of a double trough by means of a rubber-lined hose through the pipe connection shown, rises through an opening at the head of the trough, and is turned back to flow

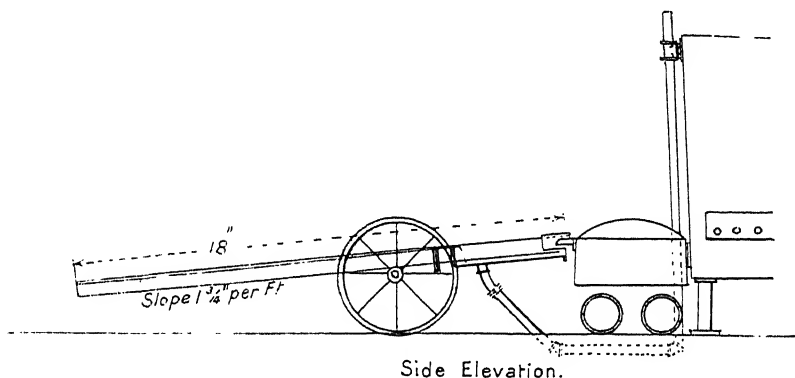


Fig. 171.

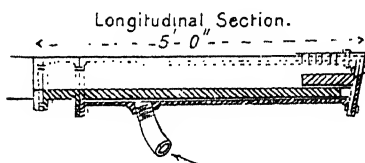


Fig. 172.



Fig. 173.

Figs. 171 to 173.—M'Arthur Granulating Trough.

over the bottom of the cast-iron inner trough by a heavy cast-iron deflector 2 inches thick. The stream of slag from the forehearth falls from a short spout upon the deflector, and is delivered in a thin layer upon the stream of water. But little steam is formed, and the slag granulations are comparatively coarse. The capacity of the apparatus with 60 gallons of water per minute, is 150 tons of slag per twenty-four hours, or, say, 1 ton of slag for 3 tons of water.

At the works of the Mond Nickel Co.,\* the slag from a furnace smelting 250 tons per twenty-four hours is granulated by a stream of water under 20-foot head; and, no fall being available, the mixed stream is forced forward

\* Hixon, *E. and M. J.*, Sept. 22, 1906, p. 553.

through a 3-inch pipe by means of a stream from a pump issuing through a nozzle of  $1\frac{1}{16}$  inches in diameter under a head of 100 lbs. per square inch.

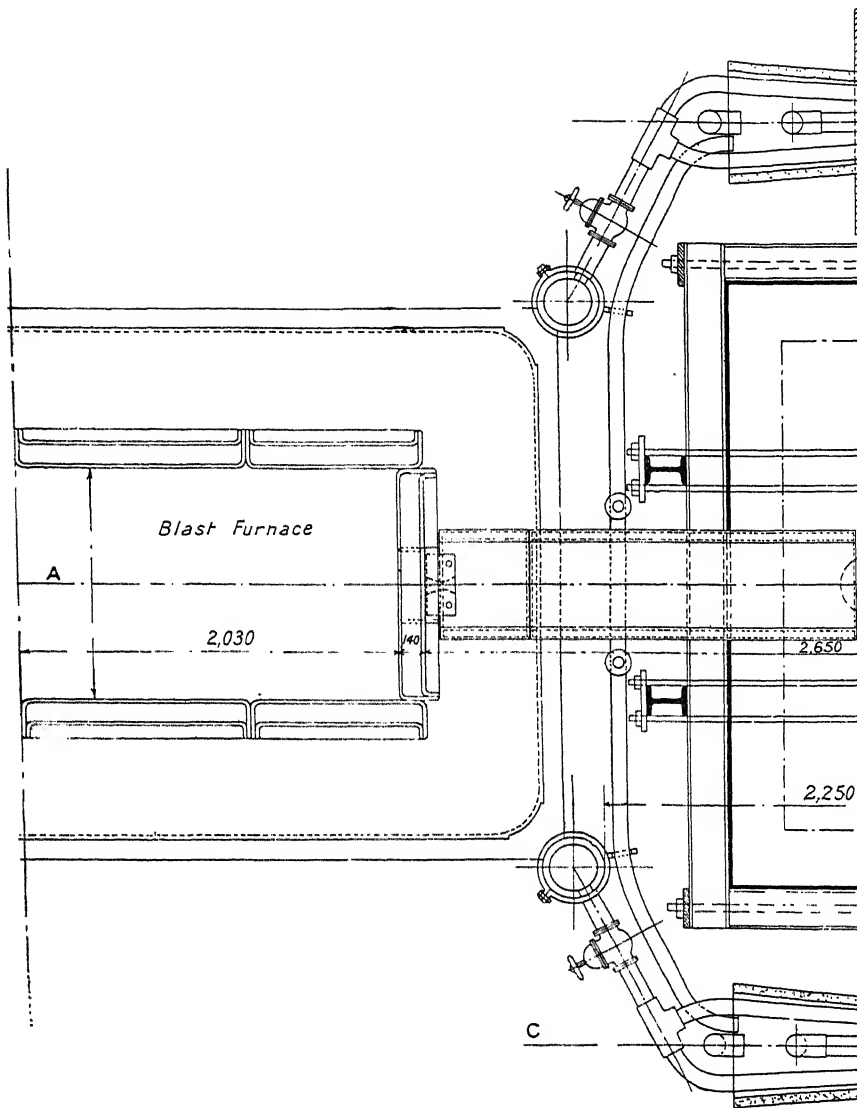
At *Laurium* the stream of slag leaves the forehearth by means of the two water-jacketed copper spouts shown in Fig. 158, which are similar in their construction to those illustrated in Figs. 162 to 166. Each of these is used alternately, and under each and at right angles to it, as shown in the plan and section of the arrangement, Figs. 174 and 175, is a heavy cast-iron trough in which the stream of slag falls upon three streams of water issuing from flattened nozzles arranged as shown in Figs. 176 and 177. These nozzles are fed only by the water from the jackets, &c., which is ordinarily found to be more than ample for the purpose, but an additional jet of water under pressure is provided for use in emergencies. The slag is broken up by the jets into granules, the average size of which is about 1 mm., the largest grain not exceeding 2 mm. Each launder discharges into a sheet-iron hopper capable of holding 8 tons, and furnished at the point with a flat slide valve moved by a rack and pinion. These hoppers take alternately the whole stream of slag and water, the latter overflowing at one side, while the former accumulates. When one hopper is full the stream of slag and water is diverted to the other, and the full one is allowed to drain for five or ten minutes by means of a perforated section at the bottom immediately above the valve: after which the valve is opened by means of a hand wheel on the pinion shaft, and the contents of the hopper are discharged into a waggon holding 8 tons. Two labourers per shift suffice to handle the slag from the furnace when smelting 300 metric tons per day, besides breaking up and loading all the matte and speiss produced.

One incidental advantage of granulating slag is that accurate sampling is facilitated, and even when no particular economical use can be found for it, the material is handled, dumped, &c., more cheaply than is hot molten slag, while the wear and tear on trucks, tracks, &c., is much lighter.

**Matte-handling.**—At some old-fashioned plants matte is still tapped from the forehearth into ordinary slag-pots, in which it is left to cool and is then turned out and broken up. The matte, however, has a great tendency to stick to the pots, even when they are clay-washed between tappings with the utmost care, and the consequent damage to pots through sledging is great, while the sledging up of the thick matte-cones involves considerable labour. A far better plan, therefore, is to use flat moulds, the depth of which is small in proportion to their area. Figs. 178 and 179 \* show the Rhodes matte-shell first introduced for this purpose by Rhodes at the *Arkansas Valley Works* (Leadville). The matte cools so quickly as to be comparatively brittle, and the risk of adhesion to the shell is at a minimum. Such small hand shells are, however, only suitable for handling a small output of matte. For a large output, whether from forehearths or furnaces, the arrangement in use at *Selby* (Cal.) † is most convenient. The shells or moulds shown in Figs. 180 to 182 are of mild steel,  $\frac{3}{8}$  inch thick, pressed into shape by hydraulic pressure, and are said to last well; they cost about 2d. per lb., or, say, 37s. 6d. per mould. Two single moulds are placed side by side upon the

\* Braden, *Trans. Amer. Inst. M.E.*, vol. xxvi., p. 38.

† *E. and M. J.*, Feb. 1, 1908, p. 254.



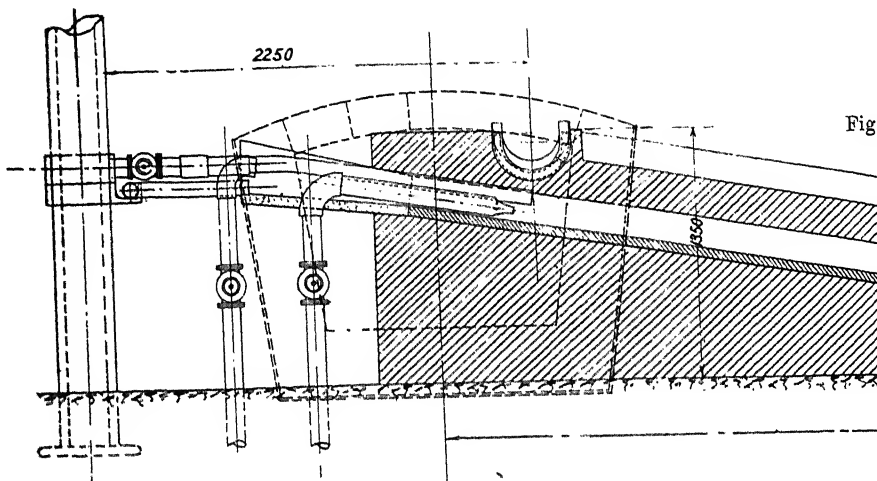


Fig. 176.—Section C D.

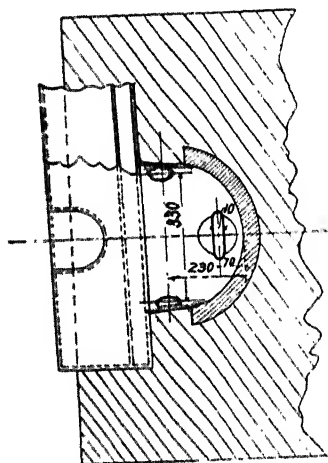


Fig. 177.—Section A B.

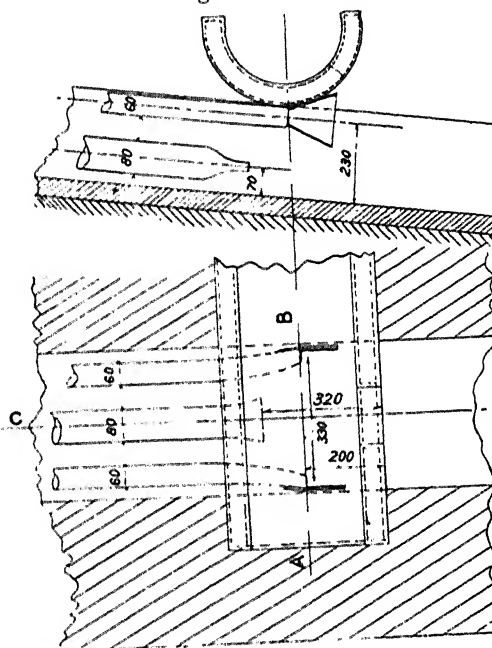
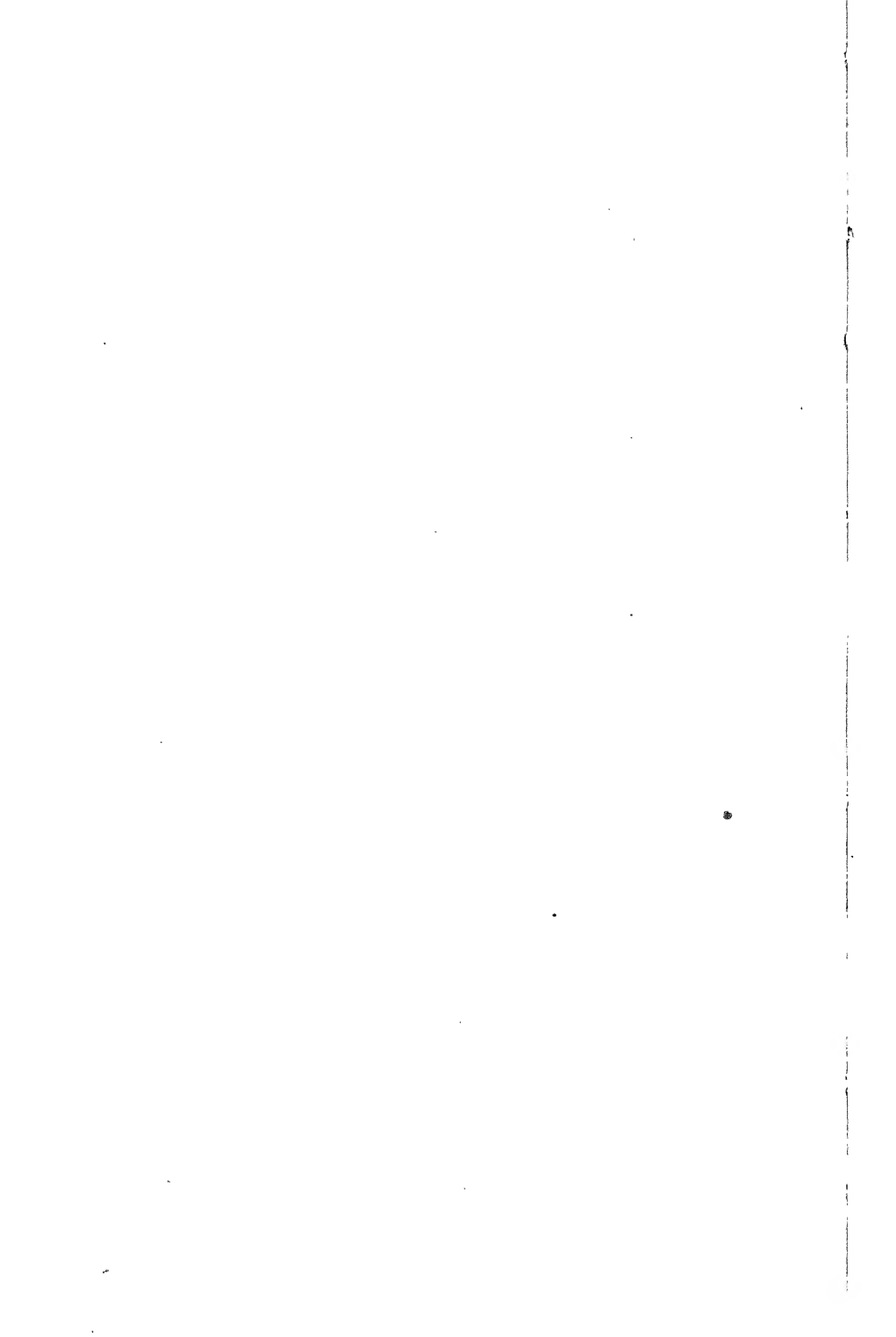
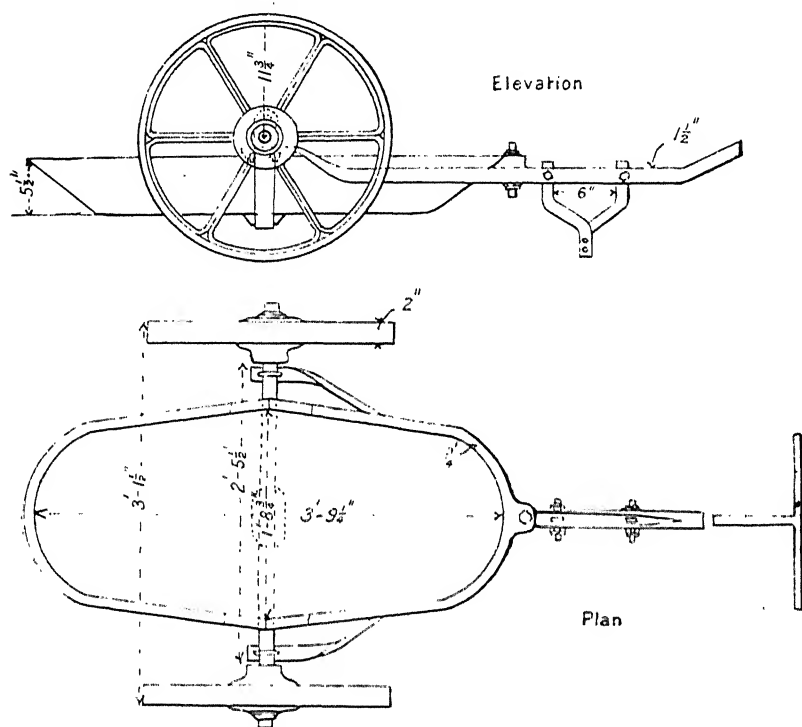


Fig. 176.—Plan.

Figs. 176 and 177.—Details of Granulating Nozzles.



trucks, shown in Figs. 183 and 184, which are pulled along by a hooked rod while the stream of matte is still running, so as to fill one mould after another;



Figs. 178 and 179.—Rhodes Matte Shell.

Fig. 180.

Fig. 182.

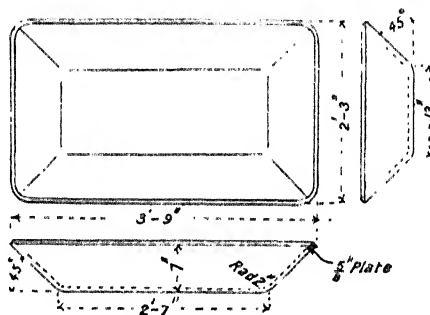


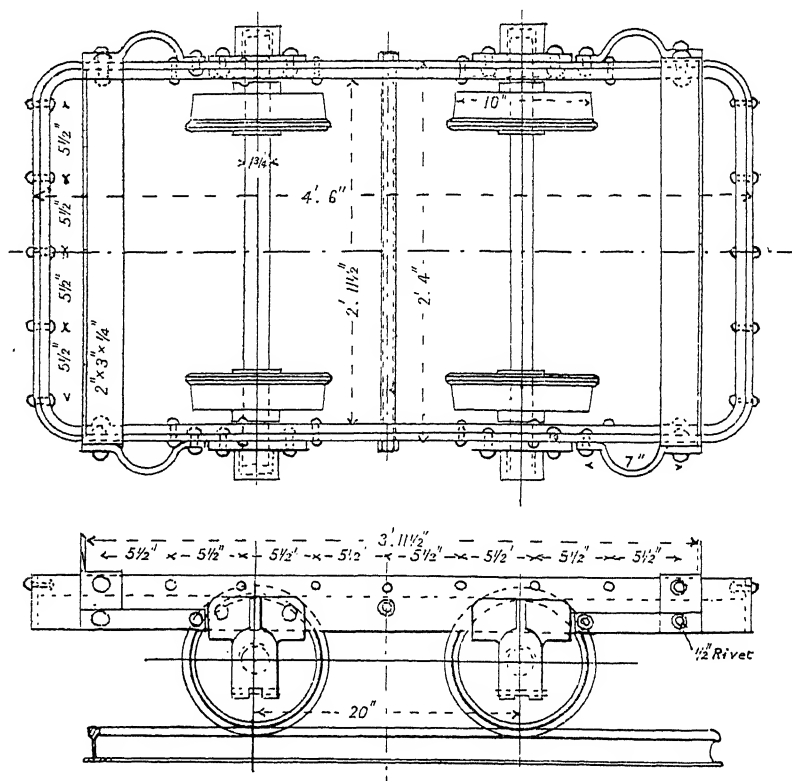
Fig. 181.

Figs. 180 to 182.—Pressed Steel Matte Moulds.

or even, if required, to fill alternately a shallow layer of an inch or two into each, which much facilitates breaking up the resulting product into small

pieces. Liability to buckling or perforation, where the stream of hot matte touches the bottom, is guarded against by placing a few large pieces of cold matte on the point affected. The author has used shallow moulds upon trucks, of a similar form, but much smaller in size, and made of cast steel,  $\frac{5}{8}$  inch thick at top and  $\frac{3}{4}$  inch thick on the bottom, and these cast steel moulds, if carefully clay-washed each time before use, last almost indefinitely.

Another convenient mode of casting matte is by means of the Kilker mould car in use at *Selby* (Cal.), and shown in Fig. 185. This consists of a rectangular truck made of I-beams carried upon cast-iron wheels with roller



Figs. 183 and 184.—Truck for Matte Moulds.

bearings, and supporting a skeleton circular cast-iron spider with sixteen arms which revolves upon a steel central pin set in the truck frame. The spider carries wedge-shaped moulds of the shape shown in the figure, each of which is hinged forward, so that it is easily dumped from a position of stable equilibrium resting upon the spider by lifting up the inner end by means of a bar. The frame is revolved under the stream of matte from the forehearth in such a way that each mould receives only a shallow layer of molten matte, which cools before the next layer comes on, so that when the moulds are dumped

the cakes are found to be formed of imperfectly adhering layers, which facilitates crushing. Each mould holds some 187 lbs. of matte, or 3,000 lbs. for the car, and the surface exposed for cooling is so great that the empty car

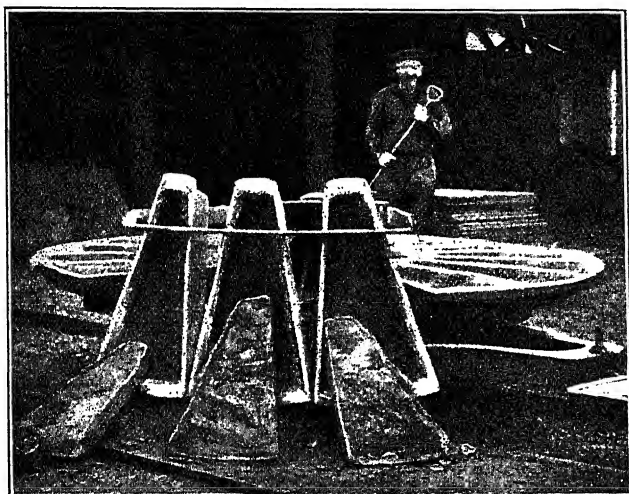


Fig. 185.

can be dumped, clay-washed, and returned for another tapping within thirty minutes if necessary; on account of the thinness of the layers cooled at one time, the wear and tear on the moulds is very small.



## CHAPTER IX.

### BLAST-FURNACE LEAD SMELTING—PRACTICE.

#### CALCULATION OF CHARGE.

**Analyses of Ores and Slags.\***—The first step towards making up a furnace charge is to ascertain the composition of the ores, especially as regards  $\text{SiO}_2$ ,  $\text{FeO}$ ,  $\text{CaO}$ , and  $\text{ZnO}$ , and, frequently also,  $\text{Al}_2\text{O}_3$ ,  $\text{BaO}$ ,  $\text{MgO}$ , and  $\text{S}$ . The methods used should be fairly accurate and at the same time rapid, for otherwise it becomes impossible to get analyses done within twenty-four hours after taking the sample, as should always be the case. As regards slags, rapidity is of vital importance, for the removal of some trouble connected with the working of the furnace frequently depends on ascertaining the composition of the slag being produced, and in this case a slag analysis to be of any use should be completed within a couple of hours or so. In order to secure this rapidity it is important to be able to decompose the slag with acid quickly and completely without a preliminary fusion. This can be always done, provided the slag sample is in a vitreous condition and very finely pulverised. Slag samples should be always taken by inserting an iron rod or small ladle into the potful of molten slag several inches deep, removing, and instantly plunging into a deep vessel of cold water, by which means the silicates are prevented from crystallising and remain in the unstable vitreous condition of solid solution.

In order to fulfil this essential condition of rapidity, it will be understood that the methods employed differ somewhat from those in use in ordinary analytical laboratories, and a short outline of some of them which are in common use will be found in the last section of this work, Chap. xix.

**Calculating the Charge.†**—It will be well to consider the most complicated case—viz., that of a custom works treating ores of very varying composition, including a large proportion of siliceous “dry” silver ores (*i.e.*, ores containing only a few per cent. of lead). At such works the ores, as received, are classified according to their composition, those which are regular supplies being mostly unloaded into stock bins, while odd lots (after sampling) are generally made up into large “beds” of several thousand tons. A classification frequently adopted is the following:—To roaster bins, (1) sulphide lead ores for blast-roasting and agglomeration, (2) pyritic and arsenical coppery and zincy concentrates for plain

\* v. Furman, *Manual of Practical Assaying*, N.Y., pp. 79, *et seq.*; Hofmann, *Metallurgy of Lead*, 274, *et seq.*; Croasdale, *E. and M. J.*, March 11, 1893.

† v. Newhouse, *S.M.Q.*, vol. ix., p. 373; Murray, *E. and M. J.*, Aug. 13, 1887, March 5, 1892; Furman, *S.M.Q.*, vol. xiv., p. 134; Hofman, *Metallurgy of Lead*, 1906, p. 310; also a paper by the author in *Proc. Civ. Eng.*, vol. cxii., pp. 133, *et seq.*

roasting. To smelter bins, (3) carbonate lead ores, (4) iron and manganese fluxing ores, (5) high grade silver ores, mostly siliceous, (6) odd lots, calcareous, barytic, argillaceous, ferruginous lead carbonates, &c., all of which are made up into "beds" together with Nos. 3 and 4. Sometimes many more classes are made, but the general custom is to keep as few as possible separate classes, and mix up almost everything in large "beds."

The first consideration must be whether there is or is not sufficient lead for separating from the other ingredients of the ore and for collecting the silver. There is no defined upper limit to the proportion of lead, for even pure roasted galena ores can be smelted in a blast furnace (though with heavy volatilisation losses); practically, however, the ore charge rarely, if ever, contains over 40 per cent. The lowest limit for good work on clean ores is about 8 per cent. by dry assay, but there is considerable loss of silver with so low a charge, unless the ores are comparatively free from zinc. With impure and refractory ores the charge should contain at least 12 to 15 per cent. lead, and, as a general average, 10 per cent. by dry assay should be reckoned, the average direct yield being about 85 per cent. of the lead present. The amount of silver in the ore should be such as to yield a "bullion" or work-lead of not more than 300 ozs. per ton. In Europe where "dry" silver ores are comparatively scarce the work-lead produced, except at Freiberg, rarely contains even one-half of the above amount. In the United States and Mexico, however, dry ores are so abundant that lead in blast-furnace smelting is regarded more as a vehicle for the collection of the precious metals than as one of the two principal metallic constituents of the ores, and, therefore, the base bullion produced occasionally runs up even to 500 ozs. Even in Denver, however, a bullion of 250 ozs. is preferred, and it is only the scarcity of good lead ores which sometimes leads to the production of a higher grade bullion.

We will assume the case of a custom works using high modern furnaces with high blast pressure, and classifying its ores received as follows:—

1. Lead sulphide ores and zinc-works residues which are blast-roasted with a small addition of lime and sometimes flue-dust and crushed ore-matte.

2. Auriferous and argentiferous pyrites, mispickel, and coppery concentrates which are roasted as thoroughly as possible in order to utilise fully their iron-fluxing value.

3. High-grade silver ores in quartzose gangue.

4. Odd lots of neutral ferruginous, calcareous, barytic, argillaceous, and other medium grade lead and silver ores, which are mixed together in beds as they come in, with the object of making as few regular classes as possible, and of diluting or swamping specially prominent features.

During the past dozen years it became customary to briquette flue-dust, and with it fine roasted and raw ores of all descriptions, with the double object of getting a good product for the furnace, and of saving losses of such fine material by dusting and even by suspension in the slag. It is further a common practice to add to the mixture for briquetting all the roasted ore-matte, and the resulting briquettes then become desirable components of a smelting charge from the point of view of their mechanical condition and flux value. In this connection reference may be made to the practice at Aguascalientes and at Laurium, described in Chap. xiii. Since the great

spread of blast-roasting, however, it is now considered even better practice to agglomerate by this method the lead-bearing fine material, and flue-dust and crushed ore-matte are also frequently added to the charge for blast-roasting. For the sake of showing the composition of a roasted pyritic ore separately in the calculation, it is supposed to form a separate ingredient of the charge, being only slightly agglomerated by ramming into the slag pots, in which it is removed from the roasting furnace: in practice, however, it would be more usual either to briquette such roasted pyrites fines, or to add them to the charge for blast-roasting and sinter the whole, utilising the iron oxide as a diluent to obviate too rapid agglomeration.

It may be further assumed that under local conditions iron flux is dear, while impure limestone is cheap.

After analysing the different ores and ascertaining what amount of each is available, the first step towards calculating a normal charge is to figure MnO as FeO, and MgO and BaO as CaO. The atomic weights of Fe and Mn being so nearly alike, the percentages of their protoxides are simply added together. Magnesia and baryta are reduced to their equivalent in CaO by multiplying by 1.4 and 0.4 respectively. When the quantity of zinc is small, ZnO is also by some reckoned as CaO ( $\approx 0.7$ ), but many metallurgists, although taking into account the total quantity present in the slag, leave it out of the calculations as regards silica-neutralising value. The analyses of the various classes of ore above described are tabulated in the following form, the analyses of the available fluxes and that of the coke-ash being added.

TABLE XX.--ASSUMED ANALYSES OF CHARGE COMPONENTS.

Material.	SiO <sub>2</sub> .	FeO MnO.	CaO MgO BaO.	Al <sub>2</sub> O <sub>3</sub> .	ZnO.	Cu.	S.	As.	Pb.	Ag.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Oz. per ton.
Sintered ore.	12	14	8	..	8	1	5	..	40	40
Roasted ..	10	50	..	..	6	3	4	2	14	20
Bed ..	28	18	10	5	4	1	5	1	16	48
Siliceous ..	80	3	3	2	1	..	1	..	3	100
Iron flux. .	10	75	..	4	..	..	..	..	..	..
Limestone. .	5	2	50	..	..	..	..	..	..	..
Coke ash. .	40	30	10	20	..	..	..	..	..	..

For the sake of simplicity the "ton" used is the American short ton of 2,000 lbs., which is commonly employed in the United States, Canada, and Australia. In Mexico and on the Continent of Europe the metric system is employed, which is even more convenient for calculation. We will assume that the available supply of these various classes of ore is approximately in the proportions given in the second column of the following table, which shows the number of lbs. of each constituent in a hypothetical preliminary trial charge supposed to be made up of the different classes of ore in the same proportion in which they are available in the stockbins, or in the regular supplies :--

TABLE XXI.—PROPORTIONATE WEIGHTS OF CHARGE COMPONENTS IN HYPOTHETICAL CHARGE.

Class of Ore.	Lbs. per Charge.	SiO <sub>2</sub> .	FeO MnO.	CaO MgO BaO.	Al <sub>2</sub> O <sub>3</sub> .	ZnO.	Cu.	S.	As.	Ph.	Ag.
Sintered, .	200	Lbs. 24	Lbs. 28	16	Lbs. ..	Lbs. 16	Lbs. 2	Lbs. 10	Lbs. ..	Lbs. 80	Oz. per charge. 4·00
Roasted, .	100	10	50	..	..	6	3	4	2	14	1·00
Bed, .	300	84	54	30	15	12	3	15	3	48	7·20
Siliceous, .	100	80	3	3	2	1	..	1	..	3	5·00
Total, .	700	198	135	49	17	35	8	30	5	145	17·20
Average, .	..	28·3	19·3	7·0	2·43	5·0	..	..	..	..	..

From a consideration of the above totals, seeing that limestone is cheaper than iron flux, and the zinc comparatively low in amount, it is evident that a fairly high ratio of lime to iron will be most economical; yet the combined amount of zinc and alumina is such as to render it inadvisable to aim at a very high ratio of lime to iron or at a ratio of oxygen in bases to oxygen in silica of more than 1 : 1·5. The type slag, No. 19, with a ratio of SiO<sub>2</sub> 36, FeO 31, CaO 26, will, therefore, be a good one. When making a slag so comparatively high in lime, and, therefore, of high melting point, in modern high furnaces, it may be reckoned that nearly two-thirds of the sulphur not appropriated by the copper will be eliminated as SO<sub>2</sub> (and SO<sub>3</sub>) including the small portion eliminated in the slag.

We will assume that the furnace is to take half-charges of 1,500 lbs. on each side alternately, out of which 200 lbs. or 13 per cent. shall be returned slag, and that each ore charge of 1,500 lbs. is to be smelted with a fuel charge of 250 lbs. of inferior coke containing 15 per cent. ash.

The complete calculation of a charge requires five separate computations :—

1. The copper present being all calculated to Cu<sub>2</sub>S, how much metallic iron will be required to form FeS with one-third of the remaining sulphur ?
2. How much metallic iron will be required to form Fe<sub>3</sub>As<sub>2</sub> \* with the arsenic present ?
3. How much iron ore is required to yield the above amounts of metallic iron ?
4. How much flux is required for the coke ash ?
5. How much flux is required for the ore mixture ?

(1) Sulphur.—700 lbs. total ore contain 8 lbs. total Cu.

$$S : Cu_2 :: 32 : 126·8 \dagger = 0·2524 : 1$$

$$8 \text{ lbs. Cu} \times 0·2524 = 2·02 \text{ lbs. S required to form Cu}_2\text{S.}$$

\* In practice, with the proportion of arsenic supposed, little or no speiss would ordinarily be formed, except when reduction was exceptionally high. When pushing the furnace with a high blast to get through as much tonnage as possible, as is usually the case in custom work, so small a proportion of arsenic in the charge would ordinarily be driven into the flue-dust, except such as entered the work lead and matte, the slag, of course, becoming correspondingly more ferruginous.

† According to the atomic weights.



From equation (2), substituting the figures of the analysis, we have

$$30 + 0.75x + 0.02y = \frac{3}{8} \cdot (40 + 0.10x + 0.05y),$$

And, simplifying, we get

$$x = 0.031y + 5.94. \quad \text{II.}$$

Then from I. = II. we get

$$\left. \begin{array}{l} y = 33.3 \text{ lbs. limestone} \\ x = 7.0 \text{ lbs. iron ore} \end{array} \right\} \text{required to flux 100 lbs. coke ash.}$$

(5) Fluxes for Ore Mixture.—

Pounds.	Material.	SiO <sub>2</sub> . Per cent.	FeO. Per cent.	CaO. Per cent.
100	Ore mixture,	28.3	19.3	7.0
<i>x</i>	Iron ore,	10	75	..
<i>y</i>	Limestone,	5	2	50

$$(1) \text{FeO} = \frac{a}{b} \cdot \text{CaO.}$$

$$19.3 + 0.75x + 0.02y = \frac{3}{8} \cdot (7.0 + 0.50y).$$

Simplifying, we get

$$x = 0.872y - 13.15. \quad \text{III.}$$

$$(2) \text{FeO} = \frac{a}{c} \cdot \text{SiO}_2.$$

$$19.3 + 0.75x + 0.02y = \frac{3}{8} \cdot (28.3 + 0.10x + 0.05y).$$

Simplifying,

$$x = 0.035y + 7.63. \quad \text{IV.}$$

Then, from III. = IV.

$$\left. \begin{array}{l} y = 24.83 \text{ lbs. limestone} \\ x = 8.50 \text{ lbs. iron ore} \end{array} \right\} \text{to flux 100 lbs. ore mixture.}$$

Thus to flux 700 lbs. of ore there will be required 59.5 lbs. of iron ore and 173.8 lbs. of limestone, making with the ore itself 933 lbs.

We have already towards our charge the fluxes for the coke ash. 250 lbs. of coke at 15 per cent. ash gives 37.5 lbs. ash, which requires

Limestone,	12.5 lbs.
Iron ore,	2.5 „
	<hr/> 15 lbs.

Also the slag charge is 

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200 „  
215 lbs.

The difference between 1,500 lbs. and 215 lbs. is 1,285 lbs., which must be the combined weight of the ore and its fluxes in the charge.

Now, 700 lbs. of ore require in all of fluxes:—

To flux silica as above, Limestone,	174 lbs.	
„ „ „ Iron ore,	59 „	} 115 lbs.
To form „speiss and matte, „	56 „	
Add weight of ore,	700 „	
Weight of ore and its fluxes,	<hr/> 989 ..	

$939x = 1,285$  gives us  $x = 1.3$  as the coefficient by which to multiply the above figures, and we have for our half-charge :—

TABLE XXII.—COMPOSITION OF FURNACE CHARGE.

	Fluxes Lbs.		Total.	In Practice.
	For Ore.	For Coke Ash.		
Roasted ore, . . . . .	910 lbs.	total.	260	260
Slagged „ . . . . .			130	130
Bed „ . . . . .			390	390
Siliceous „ . . . . .			130	130
Iron „ . . . . .	149	2.5	152	150
Limestone, . . . . .	226.5	12.5	238	240
Slag, . . . . .	..	..	200	200
			1,500	1,500

The column headed “in practice” shows the charges actually weighed out, the furnace scale beams being usually set only to 5 lbs. If any portion of the ore or flux is wet, as, for example, the siliceous or the bedded ores or the iron flux, allowance is made by adding on a certain percentage (as indicated by the result of a special moisture sample) to the weight of that class of material. The total weight of charge will then run over 1,500 lbs. The same is done in the case of wet coke. Table XXIII. shows the total number of lbs. of each constituent in the combined half charge of 1,500 lbs. ore, slag, and fluxes, and 250 lbs. coke. The total number of lbs. of slag—viz., 1,050—is found by adding together the number of lbs. of  $\text{SiO}_2$ ,  $\text{FeO}$ ,  $\text{CaO}$ ,  $\text{Al}_2\text{O}_3$ , and 90 per cent. of the  $\text{ZnO}$ , and allowing  $3\frac{1}{2}$  per cent. extra for  $\text{PbO}$ ,  $\text{S}$ , alkalis, &c. The amount of zinc going into the slag will vary a good deal, but, as a rule, from 10 to 20 per cent. of the total present will cover the amounts volatilised and carried into the matte. The coefficient for proving that the slag has been correctly calculated is obtained by  $1,050x=100$ . The amount of lead in the charge (the total charge of ore and fluxes without slag is usually referred to) is 14.5, or, including the slag also, 12.6.

The quantity of matte is obtained by supposing the  $\text{Cu}_2\text{S}$  and  $\text{FeS}$  (already determined) to form 85 per cent. of it; the lead in it may vary from 8 to 12 per cent., but will not average more than 10 per cent., while the 5 per cent. remaining will cover zinc and other substances.

The richness of matte in silver is generally calculated by supposing the lead present to be equally rich with the work-lead,\* but this is only a rough approximation, and the matte is generally found to be richer than would be indicated by this calculation. The author finds that a much closer approximation to the amount of silver carried into the matte is obtained by reckoning that both the lead and copper sulphides in the matte are equally rich with the work-lead. Such rules, however, are only approximations, and the amount of silver carried into the matte varies not only with the nature and amount of the other impurities present, but also according to the relative proportions of the two products.

\* Furman, *S. of M.Q.*, vol. xiv., p. 144.

TABLE XXIII.—TABULATED DATA FOR CHARGE CALCULATION.

[illegible]



The matte will be further referred to in the chapter on "Products."

The quantity and richness of the work-lead may be got at pretty nearly by reckoning that 3 to 5 per cent. of the lead present and 2 to 4 per cent. of the silver go into the flue-dust, while, unless some fume filtering process is adopted, even in good work with charges low in lead, some 5 to 10 per cent. of the lead and 1 to 3, or even 4, per cent. of the silver are lost in slags and in undensified fumes. These figures are, of course, based upon commercial assays, the true figures of corrected assays would frequently show much higher losses. The remaining lead and silver, less that accounted for in the matte, will be found in the work-lead. As average figures in the case under consideration we may suppose that 4 per cent. of the lead and 3 per cent. of the silver pass into the flue-dust, from which they are subsequently recovered, less the loss on re-treatment; while, say, 8 per cent. of the lead and 2 per cent. of the silver are lost. We have thus—

Wt. in Charge.	Loss in Smelting.	Flue-dust.	Matte and Work-lead.
Lead, 188.5 lbs.	— 8 per cent. = 173.4 lbs.	— 4 per cent. = 165.9 lbs.	
Silver, 22.36 ozs.	— 2 per cent. = 21.91 ozs.	— 3 per cent. = 21.24 ozs.	

The  $\text{Cu}_2\text{S}$  and  $\text{FeS}$  add up to 48.75 lbs.; as this forms 85 per cent. of the matte, the total weight of the latter will be 57.35 lbs., containing on an average, say, 10 per cent. or 6 lbs. of lead. There are left  $166 - 6 = 160$  lbs. of work-lead (about 12.3 per cent. yield on the ore and fluxes) which contains about 256 ozs. silver per short ton without allowing for that in the matte.

The matte contains 10 per cent. lead and 18 per cent. copper,\* or together 28 per cent. of material as rich in silver as the work-lead; therefore, the matte at 28 per cent. of 256 ozs. will assay about 72 ozs. per short ton, and remove about 2 ozs. of silver from each charge.

For the work-lead we have left 160 lbs. of lead, and  $21.24 - 2 = 19.24$  ozs. silver, corresponding to about 240 ozs. per short ton; the net yield of bullion being thus 12.3 per cent. on the ore and fluxes alone, or  $10\frac{1}{2}$  per cent. on the total weight of charge smelted, including slag.

No separate calculation is made for gold, since this metal is much more perfectly concentrated in the work-lead than is the case with silver, the matte containing relatively very little. So far from there being any perceptible loss in smelting gold ores, the gold obtained in bullion is usually in excess of the amount paid for or charged in the furnace books, because most of the ores and iron fluxes carry minute traces which are not taken into account.

The foregoing complete method of calculating a charge is only necessary at large works treating a variety of ores; in smaller works and in those which treat ores from only a single mine or group of mines, the calculation can often be simplified by neglecting the coke-ash, speiss, and matte, and adding a rough constant to the proportion of fluxes found for the ore charge in accordance with the result of practical experience. A simplified method of calculation on this plan is given in the paper on Smelting Processes already referred to,† and may be found useful by hard pressed metallurgists in charge of small plants. A further reason for using in practice a simpler form of

\* In practice, a matte with 18 per cent. copper would be likely to carry 15 or 16 per cent. lead, but the assumption made is sufficient to illustrate the methods of calculation employed.

† *Proc. Inst. Civ. Eng.*, vol. cxii., part ii., p. 135.

calculation is to be found in the fact that the behaviour of sulphur in the furnace, and, therefore, the proportion of matte formed, depends very largely on furnace manipulation, and particularly on the amount of blast used. If the atmosphere be distinctly reducing, as in the typical practice of the eighties and early nineties, two-thirds of the sulphur contents of the charge may enter the matte; but with a heavy blast the proportion will often not exceed one-third, as in the example cited. Even without varying the blast the production of matte varies according to the permeability of the charge, a closely-packed fine charge always yielding more matte than one of the same composition which is open and loose.

A still simpler method of calculating furnace charges consists in the use of a slide rule with slides for each base, specially graduated, so that equivalents of the different bases are measured by the same length. Such a slide rule is that devised by Wingham,\* with special reference to iron blast-furnace charges, but which can also be used for any kind of smelting, as it contains three separate silica scales—viz., for mono-, sesqui-, and bi-silicates, and base scales for lime, magnesia, alumina, ferrous oxide, and alkalis. On the back are other scales for sulphur required to form matte, iron-oxides to displace  $\text{PbO}$  and  $\text{Cu}_2\text{O}$ , limestone required to yield a given weight of lime, &c. With the aid of this simple apparatus the rough calculation of the basic or acid excess of any particular ore from its analysis is a purely mechanical operation occupying a few moments only; and, as already mentioned, this is often practically all that is necessary except in custom works where great differences exist in the nature and proportion of the bases present, and where, therefore, it is necessary to know whether the basic excess or deficiency in any given ore is an excess or deficiency of iron or of lime.

**Bullion-fall.**—When smelting lead ores chiefly or primarily for their lead contents, the bullion-fall will naturally depend upon the percentage of lead in the ores treated, and will usually be higher than in smelting ores of the precious metals with lead as a vehicle. In the latter case, on account of the cost of transport on the lead ores, it is often indispensable to economise in their use, and to run with as low a percentage of lead on the charge as possible. For best extraction with clean slags, it is advisable that the bullion-fall should amount to 10 or 11 per cent. on the charge, but clean slags can be made with a bullion-fall of only 7 per cent., provided—

(a) That the slag is of medium composition, not much above a mono-silicate in degree, with low percentages of zinc and other objectionable ingredients.

(b) That the matte-fall is not less than 5 per cent. and preferably about 8 per cent. on the charge.

(c) That proper means be provided to separate matte from slag.

**Matte-fall.**—The production of matte in lead blast furnaces was at one time regarded as a necessary evil, and a large production was considered to indicate the need of roasting more of the sulphide ore before treatment. For many years past, however, it has been generally recognised that a considerable matte-fall—namely, of no less than 5 per cent., and preferably of 7 to 10 per cent. on the charge—is a desirable thing; other things being equal, the higher the furnace the greater the proportion of matte which can

\* *Journ. Iron and Steel Institute*, 1892, part i., p. 238.

be advantageously made. The advantages of running to produce a considerable matte-fall are :—

1. The lead is better reduced, and the slags, provided the matte is properly settled out by suitable arrangements, are more free from both lead and silver with a given percentage of lead on the charge than they would be if no matte were being made, because of the fact that the large globules of matte in coalescing and settling out from the slag collect and entangle microscopic globules of bullion, which are carried by the slag in suspension, and which, by reason of their minuteness, would otherwise fail to become settled out from it.

2. A larger proportion of sulphide ores can be used direct without roasting, which saves the cost of the latter operation, since it is cheaper to burn off sulphur in a blast furnace than in a roaster.

3. The utilisation of part of the sulphur contents of the sulphides as a source of heat enables a considerable saving to be effected in the fuel consumption of the furnace.

4. Furthermore, a heavy matte-fall helps to scour out crusts, and keeps the crucible clean and hot in cases where it is necessary to run with a low percentage of lead.

There are, however, great differences in the behaviour of mattes of different composition as regards, not only their facility of separation from slag, but also their efficacy for cleaning slags; roughly, it may be said that the value of matte for cleaning slags is directly proportional to its percentage of copper (at all events, up to 15 per cent. or thereabouts), and inversely proportional to its percentage of zinc. Zincy mattes are not only light, but they require a higher temperature to melt them than do those which are rich in lead and copper, and they do not separate nearly so well from the slag, with which they form mixtures of "mushy" consistency floating on top of the lead in the crucible.

This is by no means due solely to the low specific gravity of the matte, since the difference in specific gravity of matte and slag would always be sufficient to cause settlement of the former. The imperfect separation is probably due in part to the fact that the zincy matte is at the temperature of molten slag, not perfectly fluid, and it may be in part a question of the surface tension of the matte globules floating about in the slag, which again may be influenced by the partial solubility of zinc sulphide in the slag itself. It will always be noted that when a zincy matte is produced the slag also contains a certain proportion of zinc as sulphide;\* and when copper is almost absent and the proportion of zinc high (as, for instance, when smelting Broken Hill sulphide ores without complete roasting) a low-grade zincy matte is made, which separates badly from the slag, and doubtless helps to account for the high lead losses in slags always experienced under such conditions; moreover, the zinc sulphide present in the ore greatly increases the volatilisation and fume losses. Whenever much zinc sulphide is present in ores, therefore, they should be thoroughly roasted before smelting, and only leady and coppery sulphide ores free from zinc should be smelted direct without roasting.

**Fuel Consumption.**—While a certain minimum amount of fuel is required to keep the furnace open without freezing, there is a considerable

\* The proportion may indeed run as high as 4 per cent. as formerly at Port Adelaide and at the Juliusshütte, near Goslar.

margin between this minimum amount required for merely melting the charge and that required to give proper reduction and clean slags. The use of fuel in excess of that required for the latter purpose not only involves waste, but it increases the probability of making speiss, if there is arsenic in the ore, or of making iron hearth accretions and of driving copper into the work-lead, if no arsenic be present. The heat is driven up and the top of the charge shows red, while lower down it becomes sticky from premature fusion, and so chokes the blast as to reduce the speed of smelting. The slags indeed become poor in lead, carrying only from 0.3 to 0.5 per cent., but this advantage is dearly bought at the expense of the other disadvantages, not to speak of the waste of fuel.

The indications of a slight deficiency of fuel for good work, on the other hand, are a cold top with much smoke, high speed of furnace, and slags which, even when well compounded, look cold and flow sluggishly, and carry from  $1\frac{1}{2}$  to 2 per cent. of lead, while the matte formed is usually high in lead. This condition of affairs may be partly counteracted by temporarily reducing the blast, but for its cure requires a slight addition of fuel. Both with over- and with under-reduction the lead in the crucible often shows a considerable amount of dross, in the one case from arsenic and copper, in the other from sulphur; but the slag-assay in conjunction with the other indications mentioned will explain the cause of the trouble and indicate the remedy. Up to a certain point coke fuel may be partially replaced as a source of heat by metallic sulphides, that is to say, a charge containing a high percentage of sulphur will always take a little less fuel than one containing only oxidised compounds, but only a small percentage of the fuel can be replaced in this manner. The increased proportion of matte, however, yielded by a charge high in sulphur, provided this contain copper also, and that there is but little zinc sulphide present, will to a certain extent perform the function of reduction in cleaning the slags.

When coke alone is used for fuel, as is now almost invariably the case, the consumption may vary from 11 up to 15 or 16 per cent., the lower figure being reached by plants using good coke at low altitudes, and the higher by those which use coke with a high percentage of ash and at higher altitudes. The influence of altitude upon fuel consumption has been already dealt with in the last chapter.

Generally speaking, fast running means low fuel consumption, and *vice versa*. With modern high furnaces, therefore, in which the column of charge is deep enough to absorb the heat driven up by even high blast pressures, an increase of blast may often allow of decrease in fuel, provided the decrease does not reach such a point as to chill the hearth and leave insufficient heat for proper reduction.

The right proportion of fuel is indicated not so much by the temperature of the slag and the tonnage of the furnace as by the assay of the slag and the proportions of matte (and/or speiss) produced. If, for example, the slag, while not carrying shots of lead or of matte, assays 2 per cent. of lead or upwards, and if, at the same time, the matte itself assays 20 per cent. of lead or upwards, it is evident that more iron must be reduced from the slag to clean both slag and matte, and this means an increase in the fuel consumption. If, on the other hand, both slag and matte are abnormally low in lead, while the former assays high in silver, and more particularly if, at the same time a con-

siderable quantity of speiss is being produced, it is probable that too much iron is being reduced, and the fuel consumption can be safely cut down until the matte carries 12 to 14 per cent. of lead. When the percentage of copper in the matte is fairly constant, the percentage of lead in it forms a useful measure of the degree of reduction desirable, and can be made use of in conjunction with other indications to regulate the consumption of fuel.

In connection with this matter it should be borne in mind that the lead loss in slag is by no means a measure of the total lead loss, for slags abnormally low in lead may be produced by a slow-running furnace with high fuel consumption, in which there is a distinct tendency to a sticky crucible and hot top. These conditions give rise to high fume losses owing to the temperature of the throat, and to the length of time during which the lead in the charge is exposed to volatilisation. On the other hand, a loss of 1 to 1½ per cent. of lead in the slag through imperfect reduction may be more than offset by the quick running of the furnace, which tends to diminish volatilisation of lead, owing to the shorter time occupied by the charge in its descent, and by the lower fuel consumption.

**Blast.**—In addition to the proper chemical composition of the charge and the consumption of fuel, another factor of prime importance in furnace running is the amount of blast.

The rate at which a furnace of given size will smelt a charge of given composition depends primarily on the weight of coke consumed in a given time, and this again directly on the amount of blast supplied, and on the pressure at the tuyere line, which governs the intensity of combustion. The pressure of the blast depends upon the volume supplied and upon the resistance offered by the charge to its passage, which again is a function of the height of the charge-column, and of its open or closely packed condition: it is evident that if more blast is sent into a furnace than can serve to burn up coke at the pressure caused by the resistance of the charge column, the excess will rise up through the charge, carrying heat with it, and "hot top" will result. For each given area of furnace, therefore, there is a maximum volume, and for each foot in height of charge column (supposed to be of unvarying composition) there is a maximum pressure of blast, which cannot be exceeded without bringing about persistent "hot top" with its attendant evils. Once determined, therefore, the volume of blast supplied to each furnace should be kept just inside this maximum, in order to get through as large a tonnage as possible.

#### FURNACE MANIPULATION.

This may be described under four heads—viz., "Blowing-in," "Blowing-out," Regular Smelting, and Irregularities in furnace working—all with reference to the large oblong furnaces already described.

1. **Blowing-in.**—The points to be observed in connection with this operation are, first, that the crucible of the furnace should be thoroughly dried out and heated red-hot before attempting the blowing-in proper; second, that it should be filled with hot molten lead as rapidly as possible to avoid crusting up the bottom; and, third, that the first slag which comes down should be ferruginous and as hot as possible, so that it may keep the lead

hot and not form incipient top-crusts. The various ordinary methods of attaining this end by the use of bars of lead are described in great detail in the work of Hofman,\* to which reference may be made with advantage.

When bars of lead for filling the crucible are not to be had—as has happened to the author in Mexico—small furnaces may be started direct on high-grade lead ores, preferably carbonates containing 50 per cent. or more of lead, though thoroughly roasted and agglomerated galena ore would no doubt be equally efficient. In this case, after the crucible has been heated red-hot, first with several fires of good hard wood and then with charcoal, the lead-well and syphon-tap are thoroughly heated, and then a clay plug is put in the latter and the well is piled high with glowing charcoal. Upon the charcoal fire in the bottom of the crucible are charged large clean pieces of coke, which are turned over and over under a light blast through a length of 2-inch pipe, and, as they ignite, more coke is added till the whole crucible is filled with a glowing mass. The breast is then put in, large sticks of selected charcoal or pine-logs are fed in to about the top of the jackets, and the furnace is then filled as rapidly as possible with alternate layers of coke and of the rich ore selected for blowing-in, to which just sufficient iron ore is added to form an easily fusible liquid slag.† As soon as the furnace is full, the flow of water through the jackets is increased and a very light blast is turned on. The lead in this way comes down quite hot and sufficiently fast to fill the crucible in a few hours, much of it no doubt liquating down before the temperature rises high enough to melt the gangue or fluxes; slag is tapped at intervals, and as soon as lead begins to run from the slag spout the lead well is cleared from charcoal, the plug in the syphon-tap is knocked out with a bar, and the furnace can be put on its normal charge. When care is taken in filling the crucible, not to get too much of the coke burnt away or clinkered, this method of blowing-in without using bars of lead can be made to give good results, in a small furnace, but the crucible always gets more or less crusted up with slag, and it is decidedly preferable to use bars of lead, if they are procurable: for modern large furnaces the use of lead is probably essential.

**2. Blowing-out.**—When a furnace is in want of repairs or gets crusted up in the crucible; when some serious accident happens to the water supply; or when fuel or ore runs short, as occasionally happens, it is necessary to blow-out. This is done in the inverse way to blowing-in—viz., by stopping the regular ore charge and substituting for it slag with iron ore until the ore left in the shaft has been smelted out. The charge is then allowed to sink; the dampers leading to the dust flues in the case of furnaces with telescopic stacks being closed, while in the case of open-top furnaces the cast-iron coverplates are put on and luted down. When the charge has sunk so that flame begins to make its appearance on top, or even very heavy lead fumes without flame, the blast is shut off, the tuyeres taken out, the slag tapped till no more will run, the tapping jacket and breast removed, and the contents of the furnace are drawn. This is done by raking them out with iron hoes into a succession of wheelbarrows, which are taken away and cooled with water, the operation

\* *Metallurgy of Lead*, New York, 1906, pp. 319-326.

† All blowing-in charges should be made ferruginous, approximating to the composition of an iron monosilicate, inasmuch as this silicate is particularly liquid, even at temperatures but slightly above its formation point.

being finished by pulling down the front jacket and raking out the remaining contents of the furnace down to the level of the bottom of the jackets. Last of all, the molten lead in the crucible is ladled out into moulds, and the bottom crusts are chipped off.

**3. Regular Smelting.**—In regular work the charges are made up either by weighed quantities of ore taken from separate bins, the analysis of which has been previously determined, or from ore beds of several thousand tons, each composed of layers of ore of different compositions approximately in suitable proportions for the furnace. The former system has been described in a paper by the author \* as employed at the El Paso Works (Tex.); the latter is the more usual system on the Continent of Europe and in most purely custom works. Most large works adopt a combination of both, stock-bins being used for certain large and regular supplies of ore, the composition of which varies only within narrow limits, while ore beds are made up with all the odd lots of ore received. In any case, fluxes are weighed separately. At some works the ore and fluxes are brought to the furnace in trucks and the fuel in wheelbarrows, more commonly both are handled in trucks; occasionally, wheelbarrows are used exclusively, though this is not to be recommended. When, as is now usual, the whole feed floor is composed of iron plates resting on I-girders, ordinary trucks can be run anywhere; but when part of the floor is of wood or stone, a good arrangement is to use scoop-shaped wrought-iron charge-barrows, or "buggies," as they are frequently called, which hold about 10 cubic feet, run on a pair of large wheels with ball-bearings, and rest while being filled on a pair of legs below the handles, that generally carry small wheels.

**Weight of Charge.**—All the items composing the charge must be accurately weighed into the truck or handbarrow. Almost invariably the multiple-beam scale is employed, with a separate beam for each kind of material composing the charge. The sliding weights on these being once adjusted, the front of the case is locked by the furnace foreman, and the beams are thrown in and out of gear one by one as required, by means of separate levers on the outside.

The cost of handling blast furnace charges in hand-barrows or buggies from the bins to the furnace may amount to from 9d. to 1s. per ton under American conditions, and with exceptionally good arrangements can be brought as low as 7d., which, however, is a minimum for high-priced labour. Where lower-priced labour is employed the cost ranges from 7d. to 10d. per ton. Under copper-smelting conditions, handling ores by means of big hopper cars dumping into bins, and filling from these by means of shoots on to suspended hopper scales, with electric transportation to the furnace and automatic dumping thereinto, the total cost from bins to furnace may be reduced to as low as 2½d.,† but a custom lead-smelting works cannot entirely eliminate shovel-work in connection with "bedding"; therefore, 5d. to 6d. may be looked upon as an ideal figure.

**Size of Charges.**—The absolute size of the charge is a matter for some consideration, for upon it, to some extent, depends the intensity of the reducing action. When carbonate or thoroughly roasted ores are to be treated and the values concentrated into bullion with as little matte as

\* "Smelting Processes," *Proc. Inst. Civ. Eng.*, vol. cxiv., part i.

† *E. and M. J.*, Aug. 19, 1903, p. 309.

possible (as at Broken Hill), a powerful reducing effect is essential, and small charges corresponding to only about 15 to 20 lbs. per square foot of section are preferred. When, however, much sulphides are to be used in the charge and a considerable production of matte is expected, the charges employed may with advantage be somewhat heavier, though never so large as in copper-matting or pyritic furnaces.

One great disadvantage of small charges is that the layers of fuel being more or less continuous and the whole charge-column being thereby rendered more permeable, there is a great tendency for the fuel to become prematurely ignited, the heated zone gradually rises, and "hot top" results, with its invariable concomitant—namely, more or less chill in the smelting zone before the tuyeres, which is no longer reached by the full amount of fuel. Experience proves that persistent hot-top can be frequently cured by the simple expedient of doubling the absolute size of the ore and fuel charges, without any other alteration whatever. A heavy double ore-charge which forms a distinct layer over the whole area of the furnace no doubt acts in two ways; first, by protecting the overlying layer of fuel from premature ignition through its slow conduction of heat; and, secondly, by offering only small channels to the upward current of furnace gases which, being spread and mixed, and having their pressure equalised over the whole area of the furnace in the underlying double layer of coke, are thereupon forced to make their way through a large number of narrow channels, giving up its heat on the way, and suffering considerable reduction in velocity. Large charges, therefore, if fed evenly, tend to prevent the formation of blowholes.

**Method of Feeding**—In charging, fuel always goes into the furnace first, being followed by the ore mixture, with fluxes and slag on top; occasionally the ore, fluxes, and slag are mixed more or less intimately, but more generally ore goes in after the fuel, followed successively by fluxes and slag. Some metallurgists, following Henrich,\* have claimed that a too intimate mixture is detrimental on account of forming a fusible slag too high up in the furnace. Seeing, however, that the only effect of facilitating the formation of slag must be to increase the speed of smelting, or, in other words, the furnace capacity, it would appear to be advantageous that the admixture of the various ingredients of the charge—ore and fluxes—should be as intimate as possible. The fuel should form an even layer over the whole area of the furnace, somewhat thicker, however, in the centre than at the sides, so as to make the centre of the charge more permeable than it would otherwise be to the current of ascending gases. As a rule, this is done by feeding some of the largest pieces of coke towards the centre line of the furnace, and then spreading the remainder of the fuel evenly over the whole area.

In connection with the charge of ore and fluxes the method of feeding should be such as to counteract the natural tendency of the current of furnace gases to find its easiest course around the outside of the charge column next the jackets. *As regards each individual charge-component*, therefore, the ideal to be aimed at is, with reference to the length of the furnace, absolutely uniform distribution; and, in regard to its width, such a distribution as will leave most of the fine particles near the sides and the bulk of the coarse

\* *E. and M. J.*, Dec. 27, 1890; Jan. 6, 1891.



material towards the centre.\* The words in italics are emphasised because, taking the charge as a whole, the separation of coarse and fine would usually result in depositing most of the limestone flux towards the centre, and most of the soft leady and ferruginous material towards the sides, which is obviously undesirable. While distributing uniformly over the whole area each of the items comprising the charge, so that ore and fluxes shall be everywhere found in their proper proportions, the object to be attained is a uniform resistance over the whole area of the charge, which is indicated on the feed floor by the quiet and even emission of smoke over the whole of the surface of the charge. When it becomes evident by the formation of "blowholes" that the normal condition of the charge has been disturbed, the extra draught at such points should be checked by feeding a few extra shovelful of fine material, so as to restore the equilibrium of resistance over the whole area. A tendency for the heat to rise can usually be checked in its earlier stages by sprinkling or wetting down the charges, particularly the fuel charge. This is generally beneficial, because it tends to prevent the coke from becoming prematurely ignited, and so wasting a portion of the heat of combustion, which is all required for the actual smelting zone. When the tendency becomes more pronounced, however, it is often best checked by doubling or trebling the size of the charges for an hour or more.

In the case of hand-feeding, the desirable result of coarse material towards the centre and fine towards the sides is readily attained, more particularly in the furnaces with overhead downtake which are fed through side doors. The cost of hand-feeding, however, is quite an appreciable item in the total labour cost of smelting, and in the largest furnaces it becomes difficult to get men capable of shovelling the large tonnage per shift which such furnaces put through, not to speak of the impossibility of getting the work done with judgment when under such conditions. Within the past seven or eight years, therefore, at Helena, Pueblo, El Paso, Murray, and other places recourse has been had to the system of mechanical feeding with open-top furnaces.

**Mechanical Feeding.**—In an able paper by A. S. Dwight,† the question of mechanical feeding is exhaustively discussed, and it is pointed out that inasmuch as hand-feeding is best when it is most regular, no great difficulty should be experienced in designing a mechanical appliance which should do the work as well or even better. Hand-feeding, of course, starts with the initial advantage of a distinct layer of fuel, whereas when the furnace is fed from a hopper charge-car containing the whole of one charge, even if the fuel be carefully placed at the bottom, there will inevitably be considerable admixture of fuel and ore in the operation of dumping. The existence or non-existence of a distinct layer of fuel has now been proved to be of comparatively little importance, the vital condition for satisfactory operation appears to be the correct distribution of coarse and fine material. How this condition is attained is well illustrated in Fig. 186, which represents a cross-section of one of the furnaces at East Helena fitted with the Dwight modifications of the Hixon mechanical feed.

\* The contrary arrangement—viz., too much fines in the centre and coarse material round the outside—results in the formation of a core of only half-fused sticky material in the centre of the furnace, which extends down as far as the surface of the lead in the crucible, and often starts crusts there.

† *Trans. Amer. I.M.E.*, Nov. 1901, vol. xxxii., p. 353.

The essential features of mechanical feeding are:—

1. An open-topped furnace with a slot extending the full length.
2. A charge car extending the full length of the furnace, filled on the ground floor of the plant by means of wheelbarrows, buggies, or small waggons, and provided with a long narrow bottom-discharge corresponding with the slot on the top of the furnace.
3. Mechanical means of raising the charge car from the bin-floor level, where it is filled, to that of the feed floor, and of traversing it along the line of furnaces, so as to drop the contents over any given furnace.
4. Means provided either in the bottom of the charge car itself or in the top of the furnace for spreading the charge, so that the fine goes to the sides and the coarse to the centre, as in hand-feeding.

The most usual method of raising the filled charge car to the feed floor is by means of an incline, as shown in Fig. 74, which is a cross-section of the Pueblo plant of the Amer. Smelt. and Ref. Co. A somewhat similar arrangement is in use at East Helena (Mont.) and at Torreon (Mexico). At *El Paso*\* (Tex.) the charge car is raised to the feed floor level by means of an electric lift, and is traversed along the top of the furnaces on rails at their ends by means of an electric motor, instead of the cable used at the other plants.

As regards the charge cars, and the mode of delivering their contents, there are two principal modifications:—

1. The Hixon or East Helena charge car with a flat bottom consisting of doors opening in the centre. With cars of this form the proper distribution is effected entirely by extraneous means—namely, by fixed spreaders placed inside the feed slots of the furnace tops, as shown in Fig. 186.†

2. The Pueblo type of charge car with double-hopper bottom, which serves to direct the charge towards the walls of the furnace, and so initiates the desired correct distribution.

With both systems, transverse spreaders may still be required in the tops of the furnaces, in order to correct irregularities in the filling of the charge cars, but at East Helena such irregularities were avoided (at the cost of some extra labour) by covering over the charge car, leaving open only a long narrow

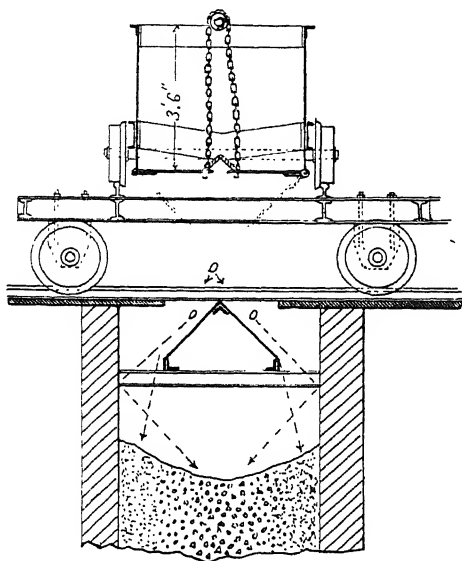


Fig. 186.—Charge-Car and Spreaders (East Helena).

\* *Private Notes*, 1902.

† From the paper by Dwight referred to, *Trans. A.I.M.E.*, 1901, vol. xxxii., p. 353.

slot above the centre line, and filling it by dumping each charge buggy in turn along the whole length of this slot. The East Helena system was somewhat more complicated in that the actual charge car had its line of motion parallel with the longer axis of the furnaces, and, therefore, had to be run upon a transfer car, in order to be traversed over the row of furnaces in a direction perpendicular to their length, while the Pueblo charge car, like that in use at Torreón (Mex.), moves entirely on a line at right angles to the longitudinal axes of the furnaces. This apparent complication, however, in actual work turned out to be of small importance. At *El Paso* \* a similar charge car of 6 tons' capacity on a transfer car or traveller is employed, the latter being traversed by an electric motor mounted on one end of the car, and taking current through an insulated third rail.

In the new form of the Pueblo charge feed devised by Howard,† instead of a car, a 4-ton feed box is employed with double hopper doors in the bottom, just the same shape as the old charge car, but handled by a travelling crane. When lowered over the furnace a set of doors on the latter are automatically opened, and after dropping the charge, the doors automatically close again. There is also a cover on the feed box itself, so as to prevent the escape of furnace gases when dropping the charge. The feed box is dropped upon a transfer car or truck for filling at ore-bin level. Figs. 187 and 188 show details of the feed box as placed on the truck for filling; Figs. 189 and 190, which are longitudinal and transverse sections respectively, show the furnace top and the manner of handling the feed box by means of the travelling crane. In these figures, AA is the travelling crane operated from P, B is the feed box resting upon the truck, C, upon which curved flat-iron brackets, I I, are reared; these cause the closing of the doors of the box when it is lowered upon the truck, and also serve as supports for it. E is the cover of the box suspended from the crane by a separate rope of such length that when the feed box is lowered into the furnace-top it remains in position, whereas, when lowered to the ore-bin floor, it remains suspended from the crane. Upon the furnace top curved guides, GG, are fastened, which guide the feed box into position and assist it to force down the hinged doors, DD; these, after the dropping of the charge and removal of the feed box, being brought back into position by the counter weights, RR, on the ends of the long levers, extensions of the hinges; L is a lever pulled by the attendant in the carriage, P, through a separate rope, which opens the hopper doors of the feed box when it is in position upon the furnace.

The advantages of this form of furnace feed are the following:—

1. The charge can be dropped from any desired height, giving either a packed charge (for coarse material) or a loose (when material is fine).
2. When required, it will drop from a much less height than the ordinary charge car, so causing less scattering of fine material and less flue-dust.
3. The furnace doors are opened and closed automatically with consequent saving of labour.
4. By the use of the cover on the feed box no fumes can escape while feeding, which makes the feed floor a more salubrious place than it usually is.

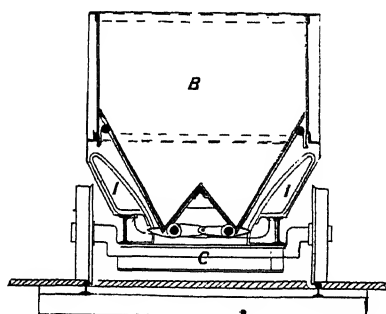
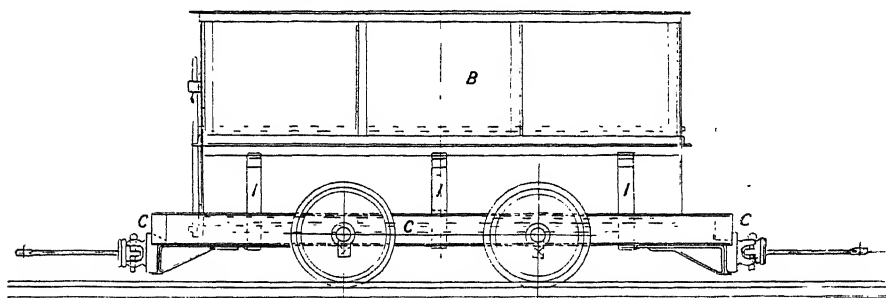
At *Mapimi* (Mex.)‡ a new system of stock bins has been installed holding

\* *Private Notes*, 1902.

† *Private Notes*, 1902.

‡ *E. and M. J.*, Aug. 22, 1908, p. 373.

10,000 tons, and provided with hydraulically operated gate doors and Anaconda track scales for quick weighing of each portion of the charge. The charge cars are formed into trains, and run up an incline to above the feed floor, where they dump into double-feed hoppers, of which two types are under trial; one type has four bottom gates operated by levers, the other type has no gates, but stands over the open-top furnace and delivers through slots in the bottom as the charge sinks.



Figs. 187 and 188.—Charging Box on Truck (Pueblo).

At *Torreón* (Mex.)\* the charges are tipped into a hopper-bottomed car, which is drawn up an incline and discharges into another hopper-bottomed charge car, which runs over the tops of the furnaces as at El Paso.

At *Bingham* (Utah)† the charging car of 5 tons' capacity, which has a V-bottom closed by trap doors, and is somewhat similar in form to the Pueblo

\* *E. and M. J.*, Jan. 20, 1908, p. 126.

† *Min. Ind.*, vol. xvi., p. 668.

charging-box, rests in a pit, where it is filled by the ordinary two-wheeled charge buggies. When full, it is raised on an incline by a rope as at Pueblo, and run over the length of the furnaces. The charge drops no less than 11 feet, and after passing through two slots closed by doors, corresponding with the bottom doors in the charging car, is distributed longitudinally by means of three heavy rails running crosswise.

At *Murray*\* (Utah) the arrangement is more like that at El Paso; the charging car is run upon an open-frame transfer truck, which is moved along

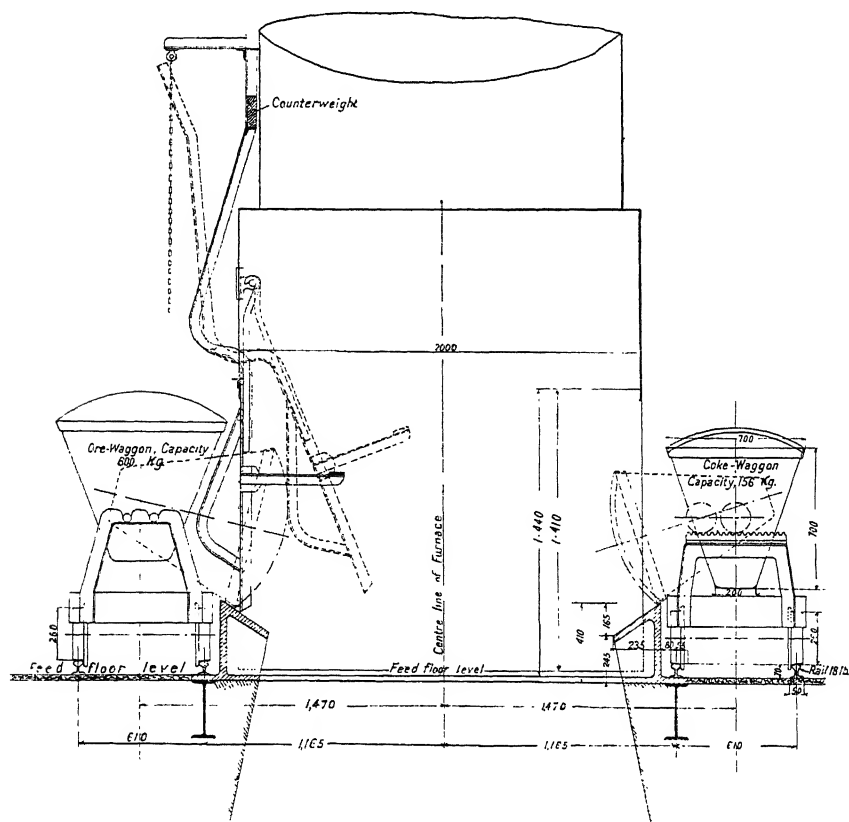
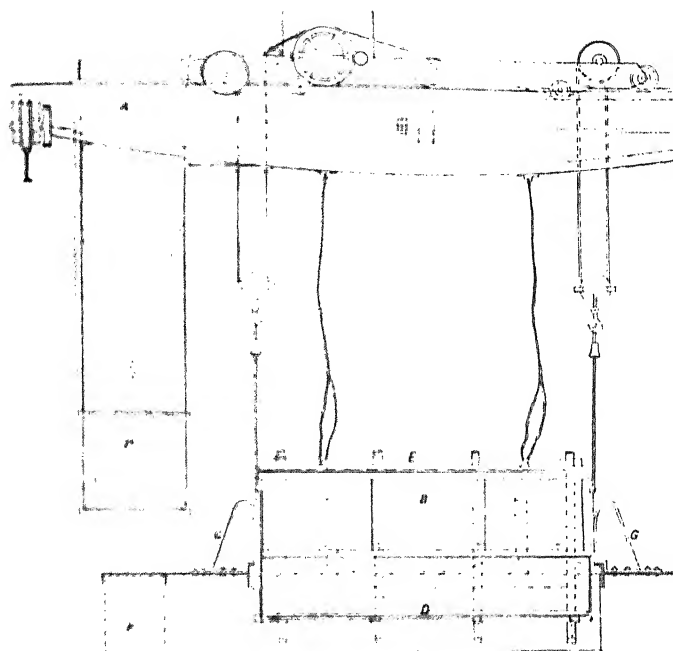


Fig. 191.—Charge Waggons and Charging Doors (Laurium). Dimensions in Millimetres.

tracks parallel to the row of furnaces by means of an electric motor. The charge drops 6 feet, and is spread by means of gable-shaped spreaders like those shown in Fig. 186.

At *Laurium* the furnaces are side-feed—that is, the take-off is above the feed floor, and connection is made with it by means of a movable charging-hood running on wheels transversely to the longer axis of the furnace, and provided with four counterbalanced doors on each side, as shown in Fig. 191.

\* *Ibid.*



Scale  
 0 1 2 3 4 5  
 10 Ft

Fig. 189 Longitudinal Section.



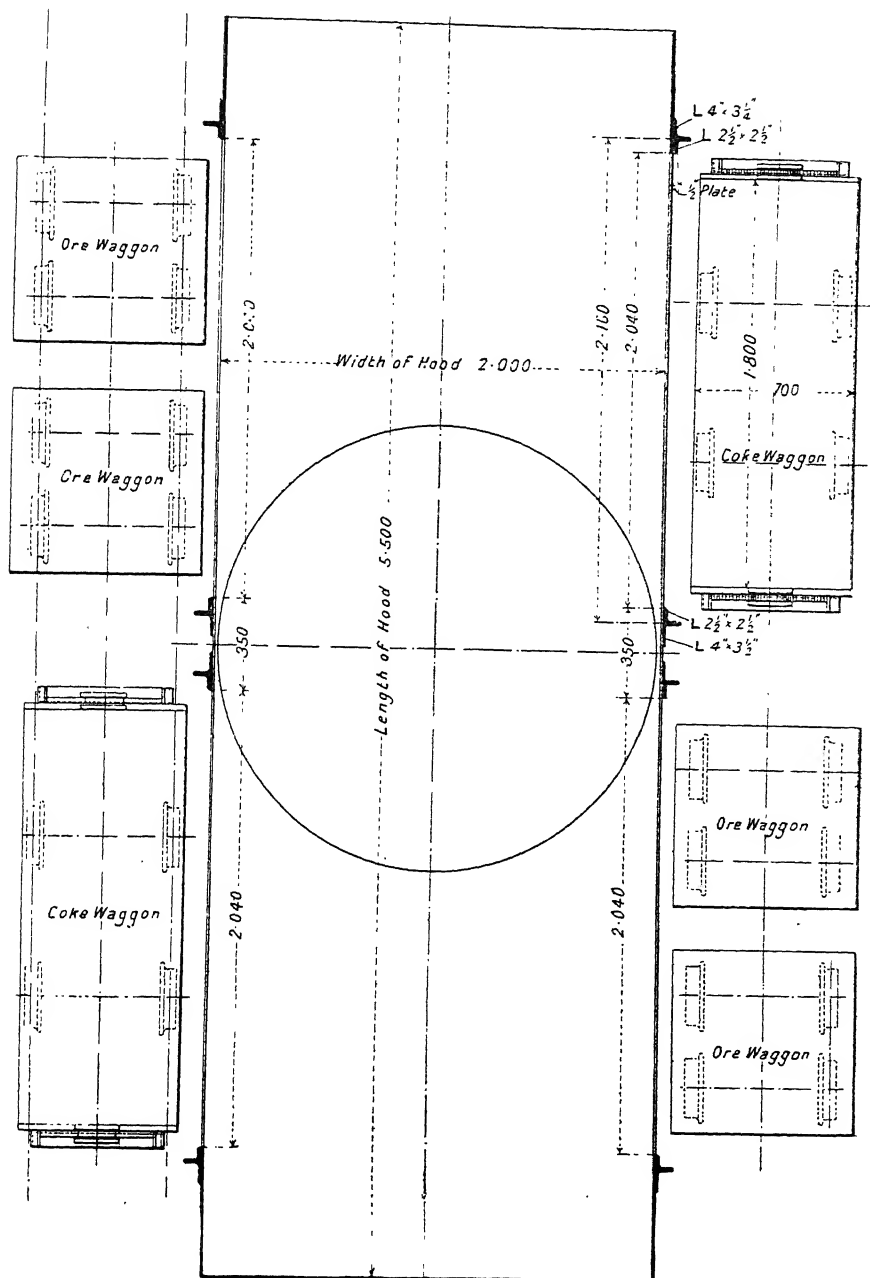


Fig. 192.—Plan of Charging Floor (Laurium).



Instead of hand-feeding, a system of mechanical feeding is employed, illustrated in Figs. 191 and 192. The ore charge is brought alternately to either side of the furnace, already mixed in four of the side-dumping trucks shown in the figure, holding 600 kilos. each. When these are tipped by hand, the counterbalanced doors open automatically, and the charge slides down the inclined cast-iron plates, shown in the figure; in the act of sliding the coarser portions gain sufficient momentum to be carried to the centre of the furnace, while the fines pile up near the sides, the overhang given to the brick-work ensuring that the finest portion shall not be directly in contact with the sides, but at some little distance from them, thus attaining the ideal condition described by Dwight without the use of any spreaders. The coke charge is similarly brought to either side of the furnace alternately in two side-tipping cars, exactly like those used for ore, but of double the length, so that each in tipping opens a pair of the counterbalanced doors.

At Laurium, owing to the fact that there are no distinctly siliceous ores and no basic fluxes, all the various ores charged containing more or less lead, lime, &c., so that each may be considered as already partially fluxed, while the mixture is self-fluxing, this system of charging gives uniformly reliable results. It is possible that with siliceous ore charges requiring the addition of large quantities of barren limestone (which in practice always turns out to be coarser than the ore, and has a tendency to run away from it), the results might not be quite so satisfactory; but it would seem as if this method, which is in some respects simpler than any of the other methods of mechanical feeding described, is worthy of imitation and of extended trial in other places where comparatively cheap labour renders it unnecessary to rely upon electric cranes, motors, &c., for moving the charge cars upon the feed floor.

#### MECHANICAL CONDITION OF THE CHARGE.

A very coarse charge varying, for instance, from lumps as large as a man's head down to about fist size, even in a high charge column, is too readily pervious to the furnace gases, and tends to cause over-fire, bad reduction, and too low a temperature in the smelting zone. The best mechanical condition for the charge mixture is about one-third in pieces from 5 down to 2 inches diameter, one-third from 2 inches down to  $\frac{1}{2}$  inch, and the remainder from  $\frac{1}{2}$  inch downwards. An excess of fine ore in the charge beyond the above proportion is attended by many inconveniences, the first being a great reduction in capacity. A furnace smelting 180 tons per day, when nearly the whole charge is from nut to fist size or larger, will put through only half to two-thirds that quantity when the proportion of fine ore rises to 50 per cent. Another and more troublesome result is a choking of the furnace, which brings about an increase of pressure, a bursting through of the fire at certain points, called blowholes, and a general upward extension of the zone of fusion, in consequence of which so much of the fuel is consumed before reaching the tuyeres that there is not sufficient left to give the slag the necessary degree of fluidity for complete separation of matte. Perhaps the worst trouble of all in the old-fashioned low furnaces (though in modern high furnaces it is neither so frequent nor so serious) is a gradual sifting-down of fine ore through interstices between the coarser lumps, assisted no doubt by the vibratory motion set up by the blast; until in extreme cases barely warm

fine ore pours into the tuyeres. Even when not reaching such an extreme, this sifting-down of fine ore is attended by the serious consequence of greatly increasing the loss of precious metals in the slag, since such fines as may reach the crucible and become carried away mechanically will be comparatively unaltered as regards their precious metal contents. Fortunately with modern high furnaces of ample area a much larger proportion of fines can be handled without serious trouble, than in the lower and smaller furnaces formerly employed.

The increased proportion of concentrates from dressing works in the ores offered, however, and the prevalence of the custom of purchasing ores through sampling works, which, in order to obtain thoroughly reliable samples, crush the whole of the ores handled, has increased the proportion of fines in the average ore mixture received by most modern custom smelting plants to far beyond the proportion of one-third, mentioned above as being the ideal for good work. Some years ago the attempt was made to overcome this difficulty, and obtain a good material for the furnace by means of slag-roasting or fusion; but this method is, as we have seen in Chap. vi., both costly and wasteful, so that it has been pretty generally abandoned. As regards comparatively rich lead ores, pot-roasting processes have successfully overcome the difficulty, and turn out a product which is in excellent condition for blast-furnace treatment. There still remain, however, the large quantities of miscellaneous ores of the precious metals, which usually form a preponderating amount of the total charge in the case of most blast-furnace smelting plants; and for these, when the proportion of fines is felt to be excessive, recourse can only be had to briquetting. Particularly is it desirable to briquette all fine ores and concentrates which are rich in silver and gold, for the mechanical losses in flue-dust and by sifting-down of fine unmelted particles into the slag become of vastly greater importance when dealing with rich material.

**Briquetting.**—Clay being objectionable as an addition to the furnace charge, lime is practically the only binder nowadays employed. Raw ore having, however, little or no chemical action upon lime, an intense pressure would be required to make coherent bricks of raw ore fines alone with the small proportion of 2 to 5 per cent. of slaked lime, which is commonly employed because a larger quantity would be uneconomical. It is, therefore, usual to mix with the raw ore fines a certain proportion of roasted ore or matte—in many cases the whole output of the roasters—and all of the flue-dust produced in the works; both of these materials contain a certain proportion of soluble sulphates, which, reacting with the lime, give rise to the formation of gypsum that acts as an effective binder.

The chief requisites for the formation of solid briquettes are thorough mixing of the ingredients and heavy pressure. Mixing is usually conducted in long horizontal pug-mills, and greater uniformity in the admixture of lime is ensured by adding this ingredient in the form of milk of lime prepared in a separate small mill. Where a very large proportion of the mixture consists of flue-dust, or roasted lead ores, heavy pressure is not essential, and fairly coherent bricks may be obtained by the use of a soft mud-brick machine of auger type as at *Laurium* (v. Chap. xiii.). Where, however, from one-third to one-half or more of the mixture consists of raw ore fines it is essential to employ a heavy pressure, if the briquettes are to be sufficiently firm to stand handling almost at once, and charging into the furnace after only a few days'

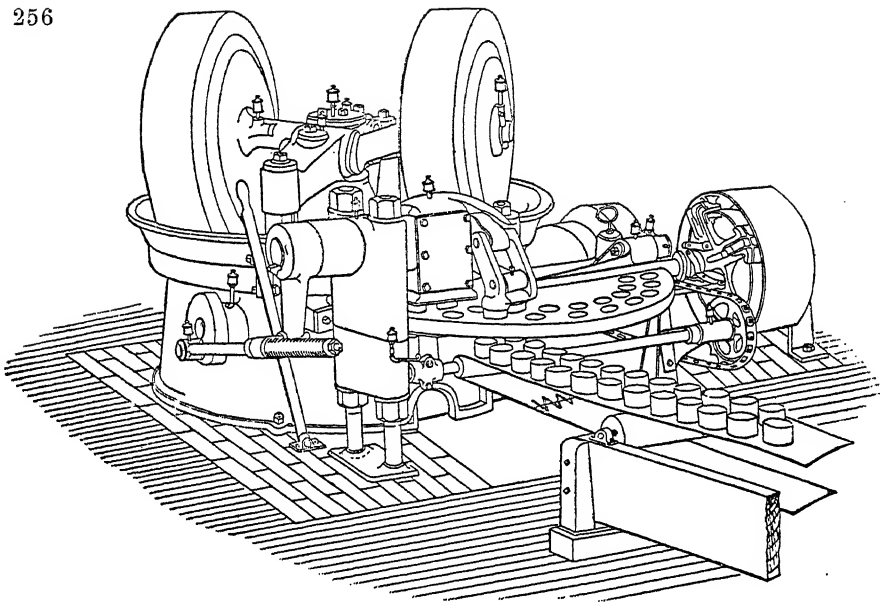


Fig. 193.—Elevation.

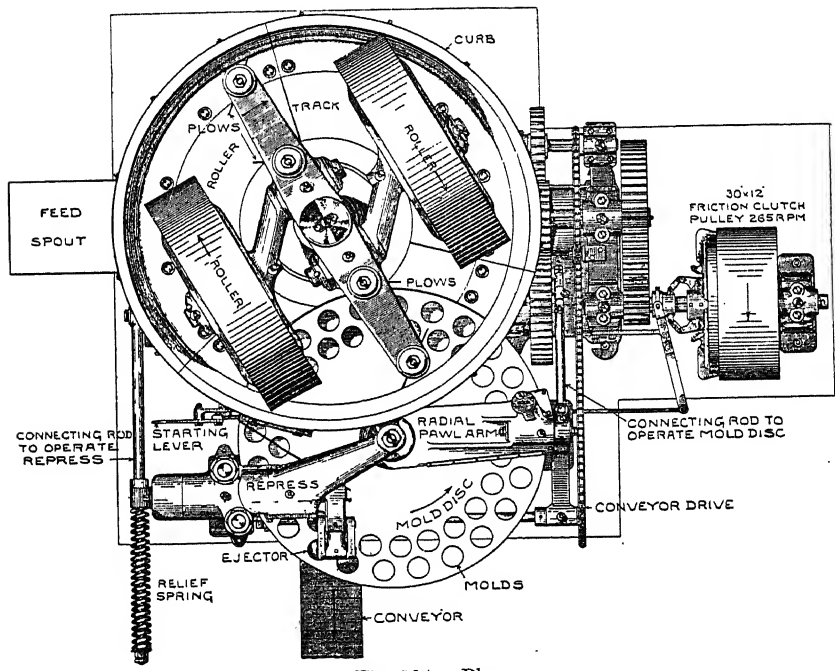


Fig. 194.—Plan.

Figs. 193 and 194.—White Mineral Press.

air-drying without breaking up to any extent. Exceedingly hard bricks can be obtained by kiln-burning, in which operation a partial fritting of the siliceous ore particles by means of the leady flue-dust or roasted lead ore takes place; this practice was followed by Greenway at the Block 14 works, where the chief material to be handled was slime from the dressing works, but for briquettes composed chiefly of fine ore and flue-dust the extra hardness obtained is hardly worth the extra cost.

Usually the mixture is briquetted, not in the form of soft mud, but of semi-dry material, just damp enough to cohere slightly when pressed in the hand. Two types of machine are in general use.

The *White Mineral Press*, shown in Figs. 193 and 194, consists essentially of a pair of rollers, 48 inches diameter and 10 inches face, with renewable tyres, and weighing 5,000 lbs. each, revolving after the fashion of a Chilian mill in a pan like that of a mortar mill, but fixed, a large section of the bottom of which is removed and its place taken by a circular moulding disc, 5 feet in diameter and  $2\frac{1}{2}$  inches thick, having two concentric rows of 4-inch circular holes near its circumference, which, as it revolves, are presented in pairs successively under the path of the rollers. Revolution of the moulding disc is effected by a ratchet and pawl, so that it moves in steps, and the same mechanism actuates synchronously, first a pair of plungers, which re-press the already formed briquettes, and, secondly, another pair which deliver two of the repressed briquettes upon a conveying belt; so that at one and the same time one pair of briquettes is formed underneath the rollers, another pair is being re-pressed, while a third pair is being delivered from the mould disc. The velocity of the rollers bears a relation to that of the mould disc, such that each pair of briquettes is subjected to the pressure of the rollers more than a dozen times before passing out of the pan. In the ordinary form of machine with two concentric rows of moulds, briquettes of 4 inches diameter by  $2\frac{1}{2}$  inches high are turned out at the rate of 80 per minute, which gives a volume of 872 cubic feet per 10-hour shift, or, say, 50 to 80 tons, depending upon the weight of the material. The machine takes about 30 H.P. to drive it, and the total cost of briquetting per ton of material varies from about 2s. 6d. up to 4s. 6d. per ton. More recently a machine has been brought out with three concentric rows of moulds, the capacity of which is 50 per cent. greater.

The other system of briquetting is that of the plunger machine, which exerts a greater pressure than the mould disc type of machine already described, and for which the mixture should be somewhat drier, containing not over 7 per cent. moisture. Two of the best known types of these machines are those of the Henry S. Mould Co., and the Boyd press of the Chisholm, Boyd & White Co., the former having the plungers horizontal, and the latter vertical. In each case the plungers are moved by means of toggle joints against very powerful springs, and they are made in several sizes with from two up to six plungers, the pressure exerted running up to 150 tons on a brick.

The Mould press turns out circular briquettes somewhat like that of the White press, 3 inches diameter and 3 inches high, of which each plunger makes 25 per minute, or at the rate of about 16 tons per plunger per ten hours. The Boyd press turns out from 5,000 to 7,000 bricks per plunger per ten-hour shift, or, say, from 20 to 30 tons, but takes about 30 H.P. for a 4-mould press.

**Pressure of Blast.**—The blast-pressure should be watched by means of the gauge on the blast pipe of each furnace, which should be constantly compared with the gauge at the commencement of the blast main just outside the blower room, and the quantity of air given regulated accordingly. Some furnacemen in single-furnace plants are accustomed to treat the blast gauge as a mere means of regulating the quantity of air which they supply to the furnace, and to continually close the furnace blast-gate when the pressure rises and open it again when the pressure falls. This plan has the great disadvantage that precisely when the furnace from any cause gets clogged up and requires more blast to push through the obstacle actually less is received, while when it is smelting well and the mass of material in the shaft is in good loose condition so much blast will be put on as to drive the heat up. A better plan is to determine by repeated trial the volume of air at given pressure which a furnace of given dimensions will take when burning coke at a given rate, and, instead of continually fiddling with the blast-gate, to keep the volume of air supplied as nearly as possible constant. This is most easily done when each furnace has its own blower, for it is only necessary to run the latter at a constant rate of speed; in the usual case of a general blast main supplying a row of furnaces it is much more difficult, because an increased resistance on any one furnace automatically diminishes the volume of air passing through it. By keeping the volume of air as nearly as possible constant, the blast-gauge becomes a delicate barometer, showing the changes which are going on inside the furnace, and in conjunction with the other signs visible at the tuyeres and on the feed floor indicating the remedy to be applied.

In all furnaces, except those with continuous slag-flow, the quantity of slag in the furnace between the slag spout and tuyere levels is under the control of the tapper, who at least once or twice in a shift usually allows the slag to rise to the tuyeres, and quickly taps it out to see the quantity in the furnace, less than the normal amount indicating the formation of incipient crusts. The depth of the pool of slag and matte below the slag spout depends entirely on the height to which the lead is allowed to rise in the lead well. Every now and then it is allowed to rise higher than usual, by dipping nothing out, by which means matte and speiss rise to the level of the taphole, and can be tapped out, flushing with them any incipient crusts on the level of the lead bath; the lead is then dipped or tapped from the lead well, and comparatively clean slag can be tapped for some time. It is very important to keep the level of the lead in the well as high as possible, especially when smelting a charge low in lead, for in this way the surface of the lead in the crucible is also brought nearer the tuyeres, and the matte and speiss are not so liable to chill. Of course, the higher the level of the lead the more is the separation of matte from slag hindered, but this does not matter when one or other of the methods of outside separation described in Chap. viii. are adopted.

**Tuyeres.**—In good work these should be covered by a thin scale of slag, through which the glowing interior should be visible in spots like stars. When a tuyere appears to be black it should be "rodded" with a pointed  $\frac{3}{4}$ -inch steel bar, so as to break away the short nose which has formed, and this treatment should be repeated as often as necessary.

**Working Force.**—The number of men required round a furnace will vary very much according to the arrangements for handling the charge, for feeding,

and for handling the slag below, not to speak of the efficiency of the labour available, so that no general rule can be given. With labour of average efficiency, say that of the Continent of Europe, and no special mechanical appliances, a large modern furnace smelting, say, from 200 to 300 tons of charge per day, may require in all per twenty-four hours from 36 to 42 men, or, say, from one-fifth to one-eighth of a man per ton smelted.

**Irregularities in Furnace Work.**—Some of the irregularities in furnace work, and the modes of rectifying them, are as follows :—

1. **Dark Tuyeres with Long Noses.**—The nose is melted off by closing the valve of the tuyere in fault, so that the heat from neighbouring tuyeres can creep close up to the iron at that point. The same remedy is adopted when lumps of ore present themselves at one or two tuyeres; when stones or fine ore appear at several tuyeres at once, the remedy is to run down the furnace a few feet and fill up rapidly.

2. **"Hot-top" (over-fire, fire-tops).**—This is a very common incident, especially in low furnaces, and may be caused by irregular descent of the charges; by too much fine ore in the charge, or by its uneven distribution; by too great blast pressure, especially with a charge composed almost entirely of coarse material, or one containing even a comparatively small proportion of exceptionally coarse material; or by wall accretions. It is usually treated at first by running down the furnace a few feet and filling up as rapidly as possible with a succession of cold charges well wetted down by means of a hose, adding a little slag over the hot places. If, however, the defect persistently recurs after a repetition of this treatment, the remedy must be sought further, probably in the removal of wall or hearth accretions. Sometimes, as has been already noted, a considerable increase in the size of the charges will cure the trouble.

3. **Uneven Descent of Charges.**—This is usually accompanied by dark tuyeres on the "hanging side," and is usually the result of bad feeding. The remedy is to feed more fuel on the hanging side, and more ore-mixture on the quickly descending side, while part of the stream of water may be turned off the jackets on the hanging side. Should this not have the desired result, a heavy 1½-inch octagonal steel bar is driven down into the hanging part of the charge to loosen it and break up any half-fused masses. In charges containing much zinc it is preferable to drive down the bar at once, so as to break off any incipient accretions.

4. **Obstruction of the Syphon-Tap.**—This may be caused by (a) lead sulphide held in solution in the lead separating out on cooling; (b) copper in the lead acting similarly. A temporary remedy is to chisel out the bottom of the well containing the lead and insert into the channel a succession of thick iron bars heated red hot. The permanent remedy is a slight alteration in the charge. Sulphide in the lead indicates a lack of reducing power caused by a want of heat in the smelting zone, and may be cured by adding a little more fuel, or, sometimes, by adding more iron flux or by slightly diminishing the blast. Copper in the lead points to the necessity of adding more sulphide ore to the charge, so as to carry it into the matte. Drosses in the crucible are often accompanied by an incorrect slag, sometimes high in lead and by an irregularly sinking furnace, showing hot-top locally and "blowing through."

5. **Wall Accretions or "Hangings."**—In the charges of many modern lead furnaces, which run so high in zinc, the growth of wall accretions is at once

the most serious and the most commonly recurring of all furnace troubles. Besides contracting the furnace area and, therefore, reducing its capacity, they always cause irregular descent of the charges, and increase the amount of flue-dust formed, through the concentration of the blast over a smaller area; while they generally cause a "hot-top," and grow so fast after once starting, that if neglected on a highly zinciferous charge, they inevitably cause the furnace to be blown out. As soon as they begin to cause irregularities they must be detached, an operation known as "barring down." At some Australian works it is customary, when treating an ore-mixture high in zinc, to drive down a long steel bar in the four corners of the furnace once every twenty-four hours, with the object of detaching incipient crusts; at others, each furnace is let down several feet every two or three days, or in extreme cases even once daily, with the object of tracing more clearly the growth of these accretions. The usual place, however, for hangings to form is about a couple of feet above the jackets, from which point they rapidly grow upward.

Removal of the hangings may be accomplished either by smelting them out or by dragging and lifting, or by both methods combined. The actual barring down is done as follows:—The furnace is fed down slowly with a slackened blast till the charge has reached the top of the jackets, no lead being drawn from the well meanwhile, so that the crucible may be full of lead, which is kept hot by the coke floating on it. The blast is then stopped entirely, the tuyeres closed, the supply of water to the jackets nearly cut off, and the whole of the slag tapped out. A double charge of coke is then thrown into the furnace, with some slag and iron flux, so as to form a bed for the dislodged material to fall upon. The cutting out of the accretions is then commenced from the bottom upwards by means of square-pointed  $1\frac{1}{4}$ - to  $1\frac{1}{2}$ -inch steel bars 16 feet long, which are driven with sledges into the crust just above the jackets from the opposite side of the feed floor. As pieces of the hangings are broken off, charges of fuel, slag, and fluxes are added, if the material is to be smelted out in the usual way, until the whole of the crusts have been cut off, when smelting is again started under a light blast, keeping the level of the charges low for an hour or two. Once broken up, the hangings will frequently "smelt out" with great rapidity, and the slag needs to be tapped very fast. If a crust has formed on top of the lead it is broken up as far as possible with a steel bar through the taphole before turning on the blast, and soon smelts out, especially if a few bars of lead or a large charge of dross are thrown into the furnace on top of the crusts, so as to come down hot and help to heat up the crucible.

If it is preferred to rake out part of the crusts, this is done by cutting away part of the bricks between the jackets and the deck frame. The broken-up accretions are then raked out and sent either to the roasting furnaces, to the fuse box of a slagging roaster, or to a separate smelting reverberatory; the opening is bricked up, a few slag charges given, the furnace filled with the ordinary charge, and the blast turned on again for regular work. "Smelting out" is, however, usually preferred as being simpler and cheaper.

The smelting out of hangings is much facilitated by charging along the sides of the furnace right up against the brickwork a small quantity of salt-cake. It is, however, advisable to let down the charge as far as the top of the hangings before adding the salt-cake, inasmuch as the corrosive effect

of this substance upon the brickwork may be almost as great as upon the accretions themselves.

6. **Hearth Accretions or "Sows."**—The formation of accretions or crusts on top of the lead is indicated by the descent of the charge in jerks accompanied by sudden bursts of flame, sparks, or even puffs of smoke; and by the lead in the well becoming dark and cold, and ceasing to "play" with the blast. The position and extent of the crust can generally be determined to some extent by inserting iron rods through the slag taphole. The most usual form of crust is one which begins at the back of the furnace and gradually grows towards the front, cutting off communication with the crucible and forcing the lead to run out with the slag. The usual cause of hearth accretions is a wrongly calculated charge, and a slag analysis will generally supply the key to the trouble. It is here that the value is seen of rapid methods which give fairly accurate results in an hour or two. In the meantime all that can be done is to give a slag charge now and then, and to slacken the blast so as to give a high temperature at the tuyeres, drawing as little as possible from the lead well, so as to keep the level high, and breaking up the crust as much as possible by driving in steel bars through the taphole and in low top-feed furnaces by driving a long bar down through the mass of charge from the feed floor.

Generally, by this means and by an appropriate alteration in the composition of the charge, the crust can be smelted out, but sometimes it will not yield to such treatment, and has to be removed from the furnace bodily. To do this, the blast is stopped and the tuyeres taken out, the openings in the jackets plugged with clay, and, after passing heavy steel bars through opposite tuyeres near the front of the furnace to hold up the mass of charge, the breast or tapping jacket is taken down. The cold slag in front is chipped away, and a little of the charge raked out, balls of clay being rammed up under the front jacket to prevent the remainder from coming down. After ladling out the pool of lead, which will be found in the front of the furnace, the crust is exposed and broken up by starting a hole in it with a  $1\frac{1}{4}$ -inch square-pointed steel bar, and then wedging it up in pieces by means of heavy 2-inch bars. After breaking up as much as possible of the crust and raking it out, the clay is removed, the space in front filled with charcoal or coke, and the breast jacket replaced; after which the steel bars through the tuyere are pulled out, the clay stoppers in the tuyere holes replaced by the tuyeres, and a gentle blast is let on. The lead ladled out, together with a few more bars, is added to the top of the charge, and smelting is resumed in the normal way, except that the taphole requires to be frequently "rodded," in order to keep open the communication with the lead in the crucible.

The above described method of removing crucible crusts, not only takes a good deal of labour, but also involves much loss of time. A new method, employing dynamite, has come into use at the works of the *St. Joseph Lead Co. (Mo.)*.\* A bomb is made from a 1-inch pipe nipple about 4 inches long, closed by means of socket unions and plugs, lined with  $\frac{1}{8}$ -inch asbestos mill-board, and charged with about one-seventh of a stick of 50 or 60 per cent. dynamite, protected at both ends by plugs of loose asbestos. The iron plug at one end of the bomb is tapped for a  $\frac{1}{2}$ -inch iron rod from 4 to 7 feet long to serve as a handle, by means of which the bomb is lowered quickly through the

\* *E. and M. J.*, Oct. 17, 1908, p. 763.



syphon-tap till it rests on the bottom in the centre of the furnace. Explosion takes place in from 15 to 25 seconds, and, while not violent enough to affect the brickwork, if the charge of dynamite above mentioned be not exceeded, it always breaks through the crust, and often splashes slag into every tuyere of a large furnace. The lead in the well is suddenly expelled, carrying with it frequently fragments of accretions, and often a quantity of mushy or half-fused material. One bomb is often sufficient to restore circulation if used in time—*i.e.*, as soon as lead first makes its appearance with the slag—but in bad cases three or four shots at half-hour intervals may be necessary to effectively break up a large accretion.

**7. Cracked Jackets.**—A cracked jacket will often produce a crust on the lead; although the crack may be sometimes temporarily stopped by throwing balls of bran or of fresh horse-dung into the water feeders of the jacket, this can be only a temporary remedy. A burnt wrought-iron side jacket requires the blowing out of the furnace for its replacement, but a cracked cast-iron "section" or end jacket can be easily replaced in the following way:—First, a full stream of water is turned on, so as to cool it and thicken the crust of chilled slag lying against it, while two courses of brick are chiselled out above. Next, the blast gate on the branch blast pipe is closed, and the water supply of the cracked jacket is shut off. It is then propped up temporarily, unbolted from its fellows, and taken down, a new one being put into its place after carefully cleaning the seat from chilled slag. Should the melted slag break through the crust, it can be stopped by clay balls, and by playing on the crust for a moment with a hose, if necessary.

**Furnace Book-keeping.**—Careful account must be kept of the work done in the furnaces, debiting to them the quantities of metals in the charge as shown by the weights and assays, and crediting the products turned out. The exact form of ruling adopted for these operating books must be decided by circumstances.

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## CHAPTER X.

### BLAST-FURNACE LEAD SMELTING—PRODUCTS.

It has already been seen that the principal products of lead blast furnaces are four in number—viz., (1) **Work-Lead**; (2) **Matte**; (3) **Speiss** (not often produced); and (4) **Slag**; besides which secondary products, such as wall and hearth accretions and flue-dust, have to be taken into account.

#### 1. WORK-LEAD.

**Work-Lead, Furnace-Lead, or “Base-Bullion.”**—The lead produced in the furnace may be either ladled into moulds direct from the lead well, or tapped from it into a movable cooling pot, from which it is ladled into moulds of thin cast iron (with the name of the works in raised letters on the bottom) of the shape shown in Fig. 195. They are whitewashed to prevent sticking, and hold on an average about 110 lbs. each; they must be carefully warmed the first time of using, but the heat of the lead keeps them afterwards at the right



Fig. 195.—Mould for Lead Pigs.

temperature. At *Tarnowitz* similar moulds are fixed upon a link belt which is moved by hand power as required, the cooled bars being dropped on to a truck below the axle carrying the belt. At a large number of works the moulds are placed in a row side by side upon a long truck, which is pushed transversely across the stream of metal tapped from a cooler in which a good deal of dross is removed.

Comparing tapping the lead well with dipping from it the latter has the advantage that the level of the lead is kept more even, by which the tendency to form a crust is lessened; it is, therefore, preferable when the percentage of lead in the charge is as low as 8 per cent. and also when the lead is to be sold to independent refining works, as in that case it is advantageous to mould a good proportion of the dross together with the lead without skimming. When, however, the refinery and the smelting works belong to the same proprietors, it is generally preferable to tap the lead into a cooling pot in which the dross is skimmed off to be returned to the furnace, where its copper contents can be taken up by the matte, and from which the skimmed lead can be ladled at leisure into the moulds. The dross is usually returned direct to the blast furnace, and few analyses of it are available; two, however, will be found quoted in Table XXIV.

TABLE XXIV.—ANALYSES OF FURNACE-LEADS AND DROSSES.

Reference,	Freiberg, Saxony.	Meer- nich, Rhenish Russia.	Przibram, Bohemia.	Andreas- berg, Upper Harz.	Overpell, Belgium.	Laurium, Greece.	Port Pirie (B.H.P.), 1896.	Port Pirie 1896.	Leadville, Colo.	Trail, B.C.	FURNACE DROSS.	
	1	2	3	4	5	6	7	8	9	10	Monterey, Mexico.	Monte- poni, Sardinia.
Pb,	95.088	99.5913	97.3597	98.9105	98.78	97.38	96.5900	98.870	99.0798	98.1014	91.9181	90.00
Ag,	0.470	0.0215	0.4230	0.0930	0.14	0.15	1.0730	..	0.6112	0.8934	0.8122	..
Au,	undet.	tr.	tr.	..	tr.	0.0004	0.0007	..	0.0009	0.0122	0.0038	..
Bi,	0.019	..	0.0070	0.0195	..	0.05	none	..	tr.	undet.	undet.	..
Cu,	0.225	0.1332	0.1100	0.0347	0.19	0.38	0.8200	0.452	0.0479	0.1800	undet.	0.85
As,	1.826	..	0.2900	0.0652	0.08	0.40	0.3800	0.124	?	0.1693	2.2340	..
Sb,	0.958	0.2180	1.5240	0.3886	0.72	1.60	0.7100	0.201	0.2139	0.6076	1.3400	0.22
Sn,	1.354	..	0.2500	0.1715	..	..	none	..	tr.	0.0289	..	..
Cd,	none	..	..	tr.	..	..	..	..	tr.	..	..	..
Zn,	0.002	0.0060	0.0012	0.0016	undet.	undet.	tr.	0.049	0.0016	..	undet.	0.40
Fe,	0.007	0.0300	0.0036	0.0054	0.04	undet.	0.0200	0.0390	0.0063	0.0140	0.0521	0.99
Ni,	..	tr.	0.0015	0.0063	..	..	none	..	..	..	tr.	..
S,	0.051	undet.	0.0300	undet.	0.08	undet.	0.1400	0.177	undet.	..	0.7598	1.60
..	..	..	..	..	..	..	..	..	..	..	..	..

References.—1. Wagner's *Jahresberichte*, 1887, p. 401. 2. *Berg- und Hüttenm. Zeitung*, 1886, p. 434. 3. *Oesterr. Jahrbuch*, vol. xxxix., p. 52. 4. Hofman, *Eng. and Min. Journ.*, Oct. 21, 1893. 5. Firket, *Ann. des Mines de Belgique*, 1901, pp. 237, et seq., and *Min. Ind.*, 1903, p. 427. 6. *Private notes*, 1908. 7. *Private notes*, 1896. 8. Bayly, *Trans. Aust. I.M.E.*, 1907, vol. xii., p. 80. 9. Emmons, "Geol. and Mining Industry of Leadville," *Mon. XII. U.S. Geol. Surv.*, p. 694. 10. *Min. Industry*, vol. xiv., p. 425; 11. Hahn, *Trans. Inst. M.M.*, vol. viii., p. 273. 12. *Oesterr. Zeitschr. f. B.- u. H. W.*, 1905, p. 455, and *E. and M.J.*, Oct. 28, 1905, p. 784.

The old method of casting lead-furnace pigs—namely, by ladling from a cooling pot—is slow, and requires much labour, particularly when the output is large. Accordingly, for some years past it has become customary at large works to allow the lead well of the blast furnace to overflow into a movable cooling pot, supported by means of trunnions from a truck, so that one pot when filled can be removed to any convenient casting floor and an empty one substituted, and provided with a spout and with worm hand-tipping gear, so that its contents can be delivered either into moulds direct, or into a large drossing or casting pot, as is more common when the pigs of lead have to be shipped to an independent refining works.

At *Aguascalientes*, *Monterey*, *Torreón*, and other Mexican plants\* the tipping pots are run to a convenient central casting shed, where their contents are tipped into one of a pair of large drossing pots provided with small fire-

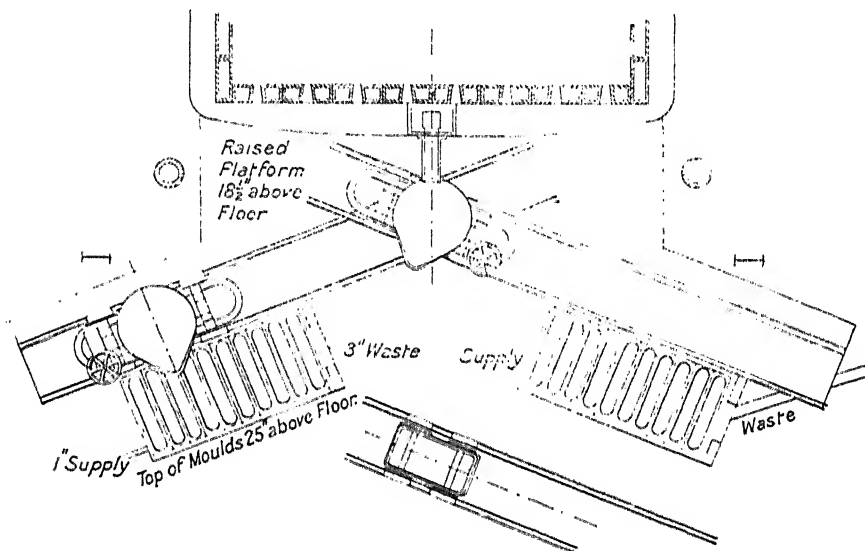


Fig. 196.—General arrangement of Lead Casting Plant (Selby, Cal.).

places and filled alternately. When one of these pots has been filled with lead at a temperature but little above its melting point, it is carefully drossed by skimming, and the contents are then cast by means of the usual Steitz syphon (for description of which see Chap. xiv.) into a row of moulds standing in a semicircle in front of the pot.

At the *National* or *No. 3 Smelter* (Monterey)\* two kettles, 6 feet 4 inches diameter by 3 feet deep, holding 20 tons of base bullion each, receive alternately the lead from all the ten furnaces. The dross is simply returned to the furnaces.

At the *Globe*, *Pueblo*, *Bingham*,† and other works a similar procedure is followed. At the first-mentioned, not only is the lead from all the blast furnaces taken in tipping pots to the 30-ton kettles to be drossed, but it is

\* *Private Notes*, 1902.

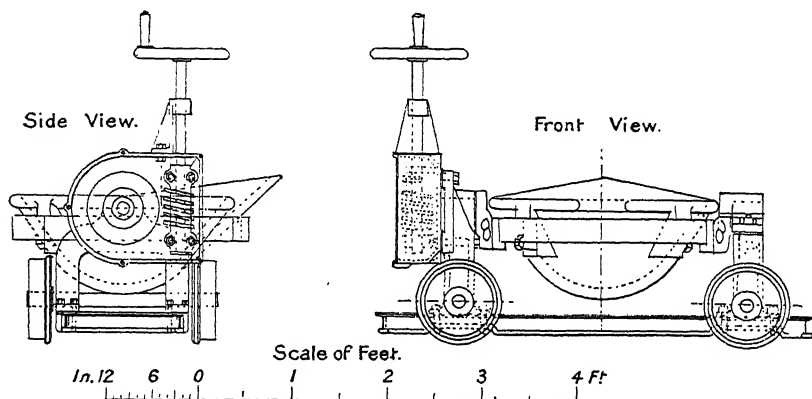
† *Min. Ind.*, vol. xvi., p. 668.

poled in these with steam for ten minutes before casting, the dross being returned to the blast furnaces.

At *Pueblo* the lead is drossed without the use of steam, and the dross, before going back to the furnaces, is put through a Howard press to free it as much as possible from lead, after which it goes back to the ore furnaces.

At *Bingham* the lead is taken from all the furnaces to three 30-ton kettles in the casting house and is drossed before casting as usual, the dross in this case being liquated in a reverberatory before going back to the furnaces.

At *Smelter No. 2, Monterey (Mex.)*,\* not only is the dross skimmed from the casting kettles liquated in a reverberatory furnace so as to get out as much lead as possible, but the liquated dross, instead of being thrown into all the ore furnaces indiscriminately, which results in circulating a good part of it continuously, is reserved and all put through with the charge of one furnace. By this means, even when copper is comparatively scarce, at least the matte from that one furnace is made high enough in copper for separate treatment, while the work-lead from the other furnaces is kept much



Figs. 197 and 198.—Lead Cooling and Casting Pot (Selby).

cleaner, and so is more easily refined. This must be pronounced far better practice than the common one of returning furnace dross indiscriminately.

The tipping pots employed for conveying the lead from the furnaces to the casting and drossing kettles vary somewhat in form and dimensions, but the general type is well illustrated by Figs. 197 and 198, which represent the tipping pots in use at *Selby* which hold ten pigs each.

In some cases, when the refinery is close at hand and under the same ownership as the smelting works, the drossing pot is dispensed with and the lead moulded direct from the tipping pots after some preliminary drossing in these.

Fig. 196 shows the general arrangement in use at *Selby (Cal.)*† for casting direct from the tipping pot into moulds. Figs. 199 and 200 show details of the water-cooled moulds in use at this plant. The moulds are cast independently, but arranged in a row of ten inside a shallow rectangular box full

\* *Private Notes*, 1902.

† Bennett, E. and M. J. vol. lxxxvi, July 11, 1908, p. 83.

of water which enters continuously through a 1-inch pipe, and overflows continuously through a 3-inch pipe. Leakage between one mould and its neighbour is prevented by the overlapping lip shown. The grooved ledge underneath the overlapping lip of the next mould was designed to hold clay or oakum packing to prevent leakage, but it was found to fill with particles of granulated lead formed as the stream struck the tops of the moulds, and so soon made a tight joint without any packing, thus preventing leakage into the water-box. The pigs are removed from the moulds with the help of a sharp steel pick, which, upon being driven into one end, lifts it sufficiently for the attendant to get his hand underneath. The capacity of the two blocks of ten moulds each is sufficient for even an abnormally large output of one large furnace.

**Composition.**—Except gold and silver, all the foreign substances present in furnace lead appear to collect during cooling in a dross which rises to the surface. When the lead is highly impure the irregular distribution of this dross (which is always poor in silver) may give rise to an extremely irregular distribution of silver in the different parts of the bar. When the lead is not under 98½ per cent. the silver is pretty

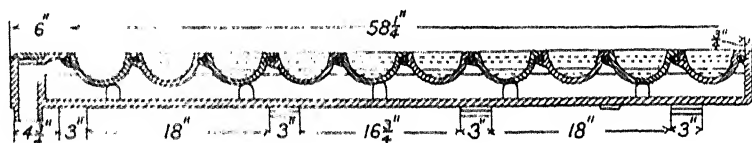


Fig. 199. Transverse Section through Water-cooled Moulds (Selby).

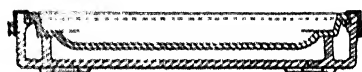


Fig. 200. Longitudinal Section Water-cooled Mould (Selby).

regularly distributed throughout it, the bottom and ends being richer than the upper portions, and the centre of the sides being the richest of all. Raht, in an elaborate article,\* shows that the distribution of silver in the bars follows the rate of cooling, and that, contrary to what might be expected from a consideration of the Pattinson process (*q.v.*) though quite in accordance with the melting points of the two metals, those parts of a bar which solidify first are the richest in silver, while the lowest silver contents are in that part which cools last—usually the centre of the upper surface. Gold is also concentrated towards the bottom of the bar, for which tendency its great density is doubtless quite as much responsible as its high melting point. The more impure the bullion the more irregular is the distribution of the precious metals.

The irregularity in composition renders necessary special precautions in sampling. A very good way is to saw the bars across and take the sawdust as sample, but the more usual way is by means of a long-handled, hollow, spherical mallet fitted with a movable hollow punch, which is simply driven

\* *Mineral Industry*, vol. iii., p. 414.

about 2 inches into each bar of lead by a single blow, the chips being forced back into the hollow mallet, whence they are removed at intervals. The usual system is to take a row of five bars face upwards and drive the punch into each in a diagonal line from the far end of the first to the near end of the fifth; then, turning the bars over, another sample is taken from each along the opposite diagonal. If the holes are correctly spaced (which after a little practice can be done with great rapidity) the sample may be approximately accurate. The chips from each truck load of bars (say 10 to 15 tons) are melted down in a plumbago crucible, well stirred with an iron rod, and poured into a mould. The resulting sample bar is drilled completely through in three or four places, or is sawn across in two places, the sawdust being taken as the final sample. More accurate spacing is ensured by marking all the bars beforehand with a suitable templet, which marks the proper position for the punch on each bar along the diagonals as explained.

Another method of sampling recently adopted at a leading American works, according to Hofman,\* consists in sawing into each bar by means of a circular saw provided with stops so as to prevent the cut going more than half-way across. The cuts are taken alternately from near the ends and from the middle, and the whole of the sawdust is taken as the sample.

As regards the accuracy of chip samples, Claudet shows † that in the case of a rich base bar of lead containing Cu 2·00, Sb 0·57, As 0·18, Fe 0·08, Ag 3·32 (=1,084 ozs. per ton), the difference between the end and the middle chip-samples on the top surface was as much as 59 ozs., or nearly 6 per cent. of the total present, and on the bottom surface 121 ozs., or nearly 11 per cent. of the total present, while the highest and lowest assays obtained from different parts of the bar were respectively 896 ozs. and 1,552 ozs., an extraordinary difference indeed. With lower grade lead bullion containing Cu 0·65 per cent., Sb 0·82 per cent., As 0·04 per cent., Fe 0·08 per cent., Sn, 0·15 per cent., Bi 0·04 per cent., Ag 150 ozs. per ton, the lowest assay chip from the top surface was 144 ozs., and the highest from the bottom, 162 ozs. Such differences show that even with due precautions to take the samples in proper places with the aid of a templet, chip-sampling is only an approximation to the truth, and accurate samples can only be obtained by sawing half-way through the bars at several points and taking the sawdust as sample.

## 2. MATTE.

**Composition.**—The mattes formed in lead smelting are mixtures of various metallic sulphides, iron almost always predominating, but replaced more or less by lead and copper, and to a smaller extent by Zn, Ni, Co, Mn, ‡ Ag, As, Sb, and Ba; other occasional constituents being Ca, Mg, and Bi. and, under exceptional circumstances, even the alkali metals § may be found in matte. At the plants treating Broken Hill ores, which are smelted rather quickly at a low temperature, and are comparatively low in iron, the matte contains a high proportion of PbS and ZnS. The amount of copper in matte depends entirely on that present in the ore, for nearly the whole of it goes into the matte. The percentage of lead in the matte depends almost entirely upon the reductive power of the furnace, which in its turn is chiefly affected

\* *Ibid.*, vol. iii., p. 425.

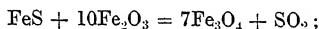
‡ *E. and M. J.*, 1900, p. 44.

† *Trans. Inst. M.M.*, vol. vi., p. 32.

§ Canby, *E. and M. J.*, Jan. 1, 1899, p. 3.

by its height, the fuel consumption, and the formation temperature of the slag. Generally speaking, the higher the furnace, and the higher the melting point of the slag, the lower will be the percentage of lead in the matte.

It is generally supposed that iron exists in mattes as  $\text{FeS}$ , and this is no doubt the case as regards the greater portion of it. The presence of magnetic iron oxide in some mattes has, however, been known for many years. Guyard,\* indeed, found 16, and even 23, per cent. of magnetite in some Leadville mattes, and supposed them to contain  $\text{Fe}_3\text{O}_4$  as a normal constituent, formed by Percy's reaction—



but there would seem to be considerable doubt as to the accuracy of his analytical methods. Hampe's published analyses of matte, however, always show oxygen as magnetite, and the author's analyses of Rio Tinto matte in 1884-5 showed from 4 to 6 per cent. of  $\text{Fe}_3\text{O}_4$  insoluble in dilute nitric acid. According to some papers by Ed. Keller,† it appears that the presence of magnetite in copper mattes is almost universal, but the lower the copper contents are, the more abundant is the magnetite. The same author believes ‡ that, at all events, in mattes from reverberatory copper smelting, magnetite is always present in mattes of lower specific gravity than its own (5·17 to 5·18); whereas, when the matte is of higher specific gravity, magnetite goes into the slags. Gibbs § has also noticed the prevalence of magnetite in reverberatory mattes, and suggests that it may be due to precipitation by some oxidising agent rather than exist as an original constituent. In the neutral atmosphere of a reverberatory furnace it is possible that  $\text{Fe}_2\text{O}_3$  in the roasted ore may be reduced to  $\text{Fe}_3\text{O}_4$  by  $\text{Fe-S}_2$  or by free S in the unroasted portion of the charge. It is, however, not so easy to account for the presence of  $\text{Fe}_3\text{O}_4$  in ordinary lead furnace mattes, and, indeed, no recent analysis of which the author has knowledge makes mention of the presence of magnetite.

As to the condition of the remaining iron it has been frequently observed that, after calculating the other metals present to their respective sulphides, there is, as a rule, insufficient sulphur left to form  $\text{FeS}$  with the remaining iron, and this is true, even if we allow for the iron present as magnetite. This excess of iron has been supposed by some to exist as a sub-sulphide, but the view of Munster and Schweder—namely, that the excess exists in the metallic form dissolved in the iron sulphide as a solid solution—seems more probable.

From 10 to 16 per cent. of metallic iron is found to exist in the blast furnace matte produced by smelting roasted pyritic ores at Zalathna (Transylvania),|| great part being in the form of distinct shots up to the size of a bean. The matte produced in matte smelting at Deadwood also contained, according to Carpenter,¶ a considerable quantity of shots of metallic iron. Both of these, however, are extreme cases of matte smelting with higher reduction than obtains in ordinary lead smelting. Besides the metallic iron present as distinct shots on cooling, Keller has found \*\* that  $\text{FeS}$  can retain in solid solution excess of Fe up to 24 per cent., and that such

\* Emmons, *Mon. XII. U.S. Geol. Survey*, 1886, p. 724.

† *Colo. S.S.M.Q.*, Dec., 1893; and *E. and M. J.*, Nov. 16, 1895.

‡ *Bull. Am. Inst.M.E.*, March, 1906, p. 282.

§ *Ibid.*, Nov. 1905, p. 1199.

|| Farbaki, *B. u. H. Zeitung*, 1894, p. 177 *et seq.*

¶ *Trans. A.I.M.E.*, vol. xxx., 1900, p. 769.

\*\* *Bull. A.I.M.E.*, March, 1906, p. 282.



mattes with excess of iron are non-magnetic, in view of which fact he suggests that the excess iron may exist in an allotropic non-magnetic form.

Pearce, on the other hand, has described \* a reverberatory matte rich in both copper and lead in which the proportion of sulphur appears to indicate that the iron present exists as  $\text{Fe}_2\text{S}_3$ . This, however, is quite exceptional, although some natural sulphides are known in which iron appears to exist in that form. It is, however, certain, as pointed out by Keller, that as regards the composition of ordinary mattes containing under 20 per cent. Pb and from 1 or 2 up to 40 or 50 per cent. Cu, the only thing constant is the percentage of sulphur, which varies between the narrow limits of 22 and 25 per cent. This percentage appears to be a sort of critical point for mattes beyond which sulphur cannot be oxidised out without iron going too. The percentage of sulphur remaining nearly constant, wide variations in the proportions of copper and lead are counterbalanced, not only by variations in the amount of iron present, but also by differences in its mode of combination. When the amount of iron is low, part may, in some cases, possibly exist as  $\text{Fe}_2\text{S}_3$ ; when high, a portion is always found as  $\text{Fe}_3\text{O}_4$ , and generally another portion also as free metallic iron. In the case of mattes very rich in lead it is possible that in some cases metallic lead may be present, simply held in solution in the excess of lead sulphide.

Lead mattes being mixtures of various metallic sulphides, their melting points vary considerably (and, indeed, as will be seen, on account of partial separation, their solidification is gradual). With the exception, however, of the non-ferruginous or little-ferruginous lead-zinc matte formerly made at Broken Hill plants, it may be taken that all ordinary lead mattes melt completely at temperatures below  $978^\circ$ , the freezing point of  $\text{FeS}$ .

Table XXV. gives, according to the best recent determinations, freezing points of some of the principal ingredients which go to make up lead mattes.

The fact that in ordinary mattes the proportion of sulphur is never sufficient to form  $\text{Cu}_2\text{S}$ ,  $\text{FeS}$  and  $\text{PbS}$  with the metals present, probably indicates that a portion of those metals exists as such, either held in solid solution in their respective sulphides, or separated out on cooling. Through the researches of Friedrich, Weidmann, Bolles, and especially of Fulton and Goodner,† it may be regarded as established that lead-copper-iron mattes after solidification may contain all or any of the following substances, and generally do contain not less than four of them, recognisable in polished sections under the microscope, viz. :—

1.  $\text{FeS}$ , capable of holding up to 5 per cent. of  $\text{Cu}_2\text{S}$  in solution, and partly dissociated at the temperature of matte formation, so as to contain a certain amount of metallic iron enclosed within it.

2. A conglomerate eutectic of  $\text{FeS}$  and  $\text{Cu}_2\text{S}$  with about 79 per cent. and 21 per cent. respectively (=17 per cent. Cu), which freezes at  $895^\circ \text{C.}$ , and which occasionally becomes dissociated into Nos. 1 and 3.

3.  $\text{Cu}_2\text{S}$ , which dissolves up to 20 or 25 per cent. of  $\text{FeS}$ , and partly dissociates at the temperature of matte formation, so as to contain a certain amount of metallic copper. This constituent is only seen in mattes high in copper, such as concentration mattes; the metallic copper appears to be

\* *Trans. A.I.M.E.*, vol. xviii., p. 65.

† *Bull. Am. I.M.E.*, Nov. 1908., pp. 959-995.

TABLE XXV.—FREEZING POINTS OF METALLIC SULPHIDES AND THEIR EUTECTICS.

Substance.	Freezing Point.	Composition of Eutectic.	Authority.
ZnS, . .	1670°	....	Friedrich v. Fulton, <i>Bull. A.I.M.E.</i> , Nov. 1908, p. 966.
Cu <sub>2</sub> S, . .	1152°	....	Hofman, <i>Trans. A.I.M.E.</i> , vol. xxxviii., p. 147.
PbS, . .	1114°	....	Friedrich, <i>Metallurgie</i> , v., 1908, p. 51.
FeS, . .	978°	....	Hofman, <i>loc. cit.</i>
Ag <sub>2</sub> S, . .	835°	....	Friedrich, <i>Metallurgie</i> , iv., 1907, p. 671.
PbS . ZnS, .	1045°	8% ZnS, 92% PbS	Fulton, <i>Bull. A.I.M.E.</i> , Nov. 1908, p. 967.
Cu <sub>2</sub> S . FeS, .	875°	21% Cu <sub>2</sub> S, 79% FeS	Röntgen, quoted by Fulton, <i>loc. cit.</i>
Cu <sub>2</sub> S . Ag <sub>2</sub> S, .	677°-1150°	No eutectic *	....
PbS . FeS, .	782°	70% PbS, 30% FeS	Weidmann, quoted by Fulton, <i>loc. cit.</i>
PbS . Cu <sub>2</sub> S, .	535°	49% PbS, 51% Cu <sub>2</sub> S	Friedrich, <i>Metallurgie</i> , iv., 1907, p. 671; confirmed by Fulton, <i>loc. cit.</i>

held in solid solution by the matte at temperatures far below its solidification point, in fact, down to 103° C., at which temperature it separates out abruptly.†

4. Metallic iron, mostly, however, in mattes low in copper and lead, and but rarely in typical lead mattes.

5. Metallic copper, only, however, in concentration mattes.

6. Zinc sulphide in crystals.

7. Enclosed slag crystals.

8. Lead sulphide, nearly pure.

9. A conglomerate eutectic of PbS and Cu<sub>2</sub>S with 49 and 51 per cent. respectively of those sulphides, and freezing at 540°.

Ordinary lead furnace mattes, containing, say, 5 to 12 per cent. copper and 10 to 16 per cent. lead, with, say, 5 per cent. zinc, do not appear to have been investigated by the above authors, but from a consideration of the constituents identified by their researches as existing in mattes with higher percentages of copper and lead, it is evident that, besides No. 6, and occasionally No. 7 of the above ingredients, such mattes will always contain as their preponderating constituent No. 1 of the above list, and, as the constituent of secondary importance, No. 9. Either No. 3 or No. 8 will be present, according as the amount of copper is greater or less than that required to form the eutectic with PbS. In addition, with mattes low in copper and high in lead, there will be a still further constituent—namely, the eutectic PbS, FeS.

\* Cu<sub>2</sub>S and Ag<sub>2</sub>S appear to be completely soluble in all proportions, and remain so when solid; the freezing points of mixtures of the two sulphides show a continuous curve with lowest point at 30 per cent. Cu<sub>2</sub>S.

† This explains the often-observed fact that rich copper mattes when broken while still hot, although quite solid, can often be seen to “sprout” and throw out filiform metallic copper.

Taking, for example, a typical lead matte with 15 per cent. Pb, 10 per cent. Cu, 5 per cent. Zn, and the balance FeS, with more or less excess Fe, its manner of cooling, according to the data and micrographical researches of Fulton and Goodner, would appear to be, according to the following stages:—\*

1. Separation of 7 per cent. ZnS at a temperature but little below the formation temperature of the mixture.

2. As the matte cooled below  $978^{\circ}\text{C}$ . gradual separation of 50.3 per cent. FeS together with free Fe, the operation being completed above  $875^{\circ}$ .

3. Separation of 2.5 per cent. of the eutectic PbS. FeS at the temperature of  $782^{\circ}$ , its freezing point.

4. Separation of 10.2 per cent. of residual FeS at some temperature below  $782^{\circ}$  and above  $535^{\circ}$ .

5. Final freezing of the eutectic PbS. Cu<sub>2</sub>S at  $535^{\circ}$ , its freezing point.

Analyses of blast-furnace mattes from lead smelting works are given in Table XXVI., a few being re-calculated to show the mode of combination of the iron.

First mattes produced in blast furnaces from a mixed ore-charge not rich in lead rarely contain more than 5 to 10 per cent. copper and 10 to 16 per cent. lead, though they may carry more lead if the fuel consumption is too low. The proportion of lead in mattes is controlled almost entirely by the slag composition and the fuel consumption. A high proportion of lead generally accompanies a high percentage of silica in the slag and almost always indicates imperfect reduction, *i.e.*, a low fuel consumption; the formation of matte unusually low in lead, on the other hand, is encouraged by basic slags, whether the excess be iron or lime, and is generally caused by over-reduction—*i.e.*, excessive fuel consumption.

**Precious Metal Contents of Lead Mattes.**—The researches of Friedrich and others summarised by Fulton and Gardner † have confirmed the opinions long held by practical metallurgists as to the relative values of various metallic sulphides in concentrating the precious metals. It was known long ago that simple ferrous sulphide has but little affinity for either gold or silver, such concentrating effect as it does possess for the former being due to the presence in it of metallic iron, which has a powerful affinity for gold, though not for silver. Metallic copper present in high-grade concentration mattes has a powerful solvent action on both silver and gold. The solvent action of zinc sulphide being practically nil, its presence in matte cannot improve the solvent power of plain iron matte. Lead sulphide, however, has a powerful affinity for silver sulphide, which it dissolves at all temperatures in proportions from 2 per cent. upwards, and it probably has some solvent action on gold. Cuprous sulphide has an exceedingly powerful affinity for silver sulphide, the two sulphides dissolving each other in all proportions, and it has also a high affinity for gold. It is, therefore, essential that for matte to be a good collector of both precious metals it should carry a considerable percentage of copper, and this was well known in practice long before it was demonstrated experimentally.

The variable gold and silver contents of lead mattes are well exemplified by the analyses given in Table XXVI.; but, in order to properly appreciate

\* For the reasoning which justifies these conclusions, reference should be made to the original paper.

† *Bull. A.I.M.E.*, Nov. 1908, pp. 960-995.

TABLE XXVI.—ANALYSES OF LEAD MATTES.

Reference,	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	Concentration Maties, Pueblo S. & R. Co.
Oker, Lower Harz.	5.09 16.81 30.53 .. .. 16.35	20 10 40 .. .. 1.5	8.01 1.56 61.00 .. 0.26 ..	11.16 1.59 41.31 1.08 .. 11.55	12.44 3.49 48.64 tr. 0.11 4.31	20.25 25.30 30.35 .. .. ..	8.00 4.85 57.0 .. 0.7 6.5	7.75 1.85 50.83 1.66 7.57 0.40	10.96 14.80 35.00 1.25 6.42 ..	12.6 10.1 35.8 .. 6.5 ..	18.0 12.4 35.2 2.2 5.4 ..	12.45 10.35 42.50 3.30 .. 3.10	17.70 9.80 33.60 2.90 .. 5.00	15.39 3.88 46.60 .. .. 6.36	16.00 43.50 13.30 1.20 .. 3.60	15.95 58.51 6.05 0.42 .. 1.24	
Ni and Co,	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Zn,	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
As,	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Sb,	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Sn,	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Ba,	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Ag,	..	0.2	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
S,	18.38	..	25.19	22.23	25.65	10.20	16.8	21.85	20.29	18.8	17.4	26.40	18.50	25.01	19.20	17.70	..
CaO,	..	..	..	0.05	0.51	..	..	..	..	..	..	0.15	1.10	..	..	..	..
MgO,	..	..	..	..	0.05	..	..	..	..	..	..	..	..	..	..	..	..
Al <sub>2</sub> O <sub>3</sub> ,	..	..	..	..	1.51	..	..	3.83	..	..	..	..	..	..	..	..	..
K <sub>2</sub> O,	..	..	..	..	0.13	..	..	..	..	..	..	..	..	..	..	..	..
Na <sub>2</sub> O,	..	..	..	..	0.10	..	..	..	..	..	..	..	..	..	..	..	..
SiO <sub>2</sub> , and insol.,	..	..	0.31	3.06	1.22	..	..	0.80	..	4.3	4.0	0.30	2.00	..	2.20	..	..
SO <sub>3</sub> ,	..	..	..	..	tr	..	..	..	..	..	..	..	..	..	..	..	..
O and C,	..	..	3.49	4.79	2.03	..	..	..	..	..	..	..	..	..	..	..	..
Au per ton oz.,	..	65	..	33	11½	30.60	6½	..	0.08	0.08	0.08	..	0.105	0.200	..	..	..
Ag per ton oz.,	..	..	..	..	..	..	..	8	86	71.9	82.0	..	85½	20.00	..	..	..
FeS,	..	..	64.13	37.70	56.84	..	..	..	..	..	..	..	..	..	..	..	..
Fe <sub>2</sub> O <sub>3</sub> ,	..	..	12.67	12.60	7.37	..	..	..	..	..	..	..	..	..	..	..	..
Fe,	..	..	11.01	9.51	7.13	..	..	..	..	..	..	..	..	..	..	..	..

Note.—Mode of combination of iron in above:—

Note.—Mode of combination of iron in above:—

References.—1, 2, and 3. Schnabel, *Handbuch der Metallhüttenkunde*, vol. i., p. 385. 4. *Oesterreichisches Jahrbuch*, vol. xxxix., p. 24. 5. Hampe, *Chemiker Zeitung*, 1882, vol. xvi., No. 28. 6. *Min. Ind.*, p. 426. 7. *Firket, Ann. des Mines de Belgique*, 1901, p. 237 (*Min. Ind.*, 1902, p. 426). 8. *Private notes*, 1905. 9. *Iles, Metallurgy of Lead*, p. 151. 12, 15, and 16. Hofman, *Metalurgy of Lead*, 1906, p. 357. 13. *Private notes*, 1902. 14. *Private communication*, 1904. 10 and 11. *Iles, Eng. and Min. Journ.*, Sept. 23, 1890.

the meaning of the figures, it is necessary to consider the precious metal contents of any given matte in conjunction with those of the work-lead produced at the same time. The matter will be referred to again in a later chapter, but the following general statements may be made here :—

1. Ordinary lead mattes produced in the ore furnace, carrying, as a rule, no free copper and very little free iron, are always poor in gold, because neither  $\text{FeS}$  nor  $\text{Cu}_2\text{S}$  and  $\text{PbS}$  have any marked affinity for that metal, and there appears to be no direct connection between the gold content of the bullion and that of the matte.

2. Of silver, however, a notable proportion is always carried into the matte, and careful investigation shows a fairly close connection between the silver in the work-lead and that in the matte, for it appears that the  $\text{PbS}$  and  $\text{Cu}_2\text{S}$  in the matte are each about as efficient in dissolving and collecting silver as is the work-lead itself. By adding together the percentages of  $\text{PbS}$  and  $\text{Cu}_2\text{S}$ , therefore, and reckoning that each will have the same silver contents as the work-lead, an approximation can be made to the precious metal contents of the matte, which, however, will generally be richer in silver than this mode of calculation would indicate.

**Treatment of Lead Mattes.**—In the first instance, lead mattes are simply returned to the ore furnace, in order to raise their percentage of copper up to from 10 to 14 per cent. They are then worked up by roasting and smelting together with a part of the ore charge, which serves the double object of extracting most of their lead and a good proportion of their silver while concentrating the copper, and securing an additional iron flux for the lead ore smelted.

**Preparation.**—Lead matte is usually obtained in the form of cakes flat on the top and having the form of the pots or moulds into which it was tapped. These cones have to be broken up by hammers and passed through a stone-breaker and rolls as a preparation for roasting in reverberatory or other furnaces. When matte is produced in large quantities in a liquid form, as is usually the case now, especially when internal or external fore-hearths, or Hcs-Rhodes reverberatories are employed, it can be advantageously granulated like slag.

At the *Arkansas Valley* (Leadville), and perhaps at some other large modern works, the molten matte tapped into hand pots is poured into the centre of a jet of water under pressure of 20 to 30 lbs. per square inch issuing from a flat fan-shaped nozzle (preferably of brass) about 3 inches by  $\frac{1}{2}$  inch. Provided the matte be sufficiently hot, the water cold, and the pressure not too low, all the matte granules will pass an 8-mesh screen and can be roasted, while the pot shells which have to be roasted or returned should not exceed 5 to 8 per cent. of the whole. At some works matte is granulated direct from the forehearths. The advantages of this practice are obvious; the only disadvantage is the (comparatively little) trouble and expense required to collect the granules from suitable tanks and load them into cars for elevation to the roasters.

There seems to be, however, a pretty general consensus of opinion that, on account probably of their polished and toughened skin, granules of low-grade matte are much harder to roast than are crushed particles of similar size. For this reason most metallurgists prefer not to granulate their matte for roasting, but to crush and roll it. Thin flat cakes are much more con-

venient for crushing than large thick masses, but when the method of matte handling adopted at the works involves casting the matte in fairly large cakes, much labour is saved in preparing these for crushing by the simple expedient of playing water sprays upon them as soon as turned out and while still red-hot, under which treatment the large masses break up, and each piece becomes fissured with innumerable joints, much facilitating its subsequent crushing.

The roasting of lead mattes has been already dealt with in Chap. v. in connection with the roasting of ores, thus *heap-roasting* as practised at Clausthal, at Penarroya, and formerly in Utah and other places in the United States, will be found referred to at p. 72; *stall-roasting*, as practised at Przibram and San Luis Potosi, at p. 73; and *kiln-roasting*, as practised at Freiberg and Lautenthal, at p. 75.

As regards roasting in reverberatory furnaces, which is the common practice, the only respects in which matte roasting differs from ordinary ore-roasting are :—

1. On account of its different physical condition, it is best to crush matte finer than ore.

2. Much more care is necessary to prevent sintering.

3. In spite of matte charges being often heavier than ore charges, they roast so much more slowly that the output of a furnace when roasting matte is always less than when roasting ore; the fuel consumption, moreover, is generally somewhat greater.

The same remarks, of course, apply to roasting matte in mechanical furnaces, and it may be added that matte being often richer in silver than the average sulphide ores that are roasted, it is advisable not to use revolving cylinder furnaces or other types subject to a heavy dust loss. Generally, therefore, hand reverberatories are still employed for roasting lead mattes in preference to any form of mechanical furnace.\*

**Concentration-Smelting of Roasted Matte.**—Formerly, when less copper was present in the ore charge of custom works than is common nowadays, it was customary to return the first matte to the blast furnace as iron flux several times, until it was concentrated in this way up to 15 per cent. copper. This practice is still followed where copper is scarce, but it is nowadays quite common for the first matte to carry 10 or 12 per cent. copper, and for this to be concentrated separately forthwith without returning to the ordinary charge.

The peculiar practice of gradual concentration followed at Clausthal, but elsewhere obsolete, will be found referred to in Chap. xi., chiefly on account of its historical interest.

At the *Aurora* Works (Ills.) a first matte containing 7 to 15 per cent. Cu, 10 to 18 per cent. Pb, and 15 to 60 ozs. Ag per ton, was roasted and returned to the ore furnace until its copper contents were over 15 per cent., when it was roasted and concentrated by smelting with siliceous copper ores; the resulting second matte, containing 10 to 15 per cent. Pb, 35 to 45 per cent. Cu, and 60 to 100 ozs. Ag per ton, was sold or worked up on a cupel hearth for gold-bearing bottoms and silver-bearing rich copper matte.

\* Table XII. gives particulars of the roasting of lead mattes in the Brown Horse-shoe and Pearce Turret types of furnace.

At the *Germania Works* (Utah)\* the first matte used to run about 10 per cent. weight of the charge, ranging from 8 to 10 per cent. Cu, and on an average 15 per cent. Pb; it was roasted and returned to the ore charge till it reached 15 to 20 per cent. Cu. It was then roasted and concentrated in separate furnaces with siliceous copper ores, producing a second matte with about 40 per cent. of copper, which was sold to copper smelters.

The usual practice nowadays is to smelt roasted matte separately from the ordinary ore charge whenever it contains 10 to 12 per cent. copper or over. Besides the roasted matte, the charge consists of all the siliceous coppery ores available (up to the fluxing capacity of the oxidised iron in the matte), and pot-shells and similar foul slag from the ore furnace. The silver contents of the ores added to the charge are kept as low as possible, inasmuch as the separation of silver from copper is comparatively costly, and only 95 per cent. of the silver present in high-grade matte is paid for by copper refiners, besides which the collection of silver by high-grade matte is not as perfect as in the ore furnace. Gold, on the other hand, being perfectly collected by matte and easily recovered from the product, it is advisable to add siliceous gold ores to the matte-concentration furnace to the extent of the fluxing capacity of the matte.

The furnace used for concentration-smelting of roasted matte is usually an ordinary lead furnace, the crucible of which has been filled with well-rammed brasque. At some works the bottom of the furnace is left inclined either from end to end or from both ends towards the centre, a slag taphole being fixed at the highest point, and another taphole at the lowest point, through which matte and lead are tapped off at intervals to settle in ordinary slag pots. At other works the whole contents of the furnace are allowed to run out into a forehearth as in copper smelting, the forehearth being provided with at least two tapholes in addition to the overflow slag spout, so as to draw off the molten products separately. Even in the former case, when most of the matte settles inside the furnace and is drawn off through its own taphole, it is good practice to provide a forehearth to receive the overflow slag in order to settle out any shots mechanically carried away.

Still another method of handling the products of concentration-smelting is that in use at *San Luis Potosi* (Mex.),† where a taphole at the side of the furnace serves for lead, while matte and slag overflow together into an Orford type forehearth, which acts like that shown in Figs. 155 to 157, but is rectangular instead of L-shaped. The matte compartment is about one-fourth the size of that from which the slag overflows, and continuous flows are maintained of both matte and slag.

The slags made in concentration-smelting are much more ferruginous than those from ore smelting; average figures may be from 30 to 33 per cent.  $\text{SiO}_2$ , 40 to 50 per cent.  $\text{FeO}$ , and 10 to 15 per cent.  $\text{CaO}$ .

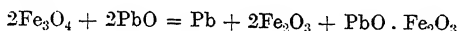
A high percentage of fuel is necessary in order to get good reduction; for example, 15 or 16 per cent. as against 11 to 13 per cent. required by the ordinary ore charge.

It is further worthy of note that the slag obtained by smelting roasted matte with siliceous ores, although low enough in silver, is usually higher in lead than an ordinary ore slag. Analysis invariably reveals the presence of  $\text{Fe}_2\text{O}_3$  in these matte slags, and it has been suggested that the oxides of

\* *Private Notes*, 1897.

† *Ibid.*, 1902.

lead and iron probably exist in the slag mutually combined as a result of the reaction of litharge on magnetic iron oxide present in the roasted matte, the reaction being as follows :—



the resulting ferrite of lead being difficult to reduce. Whatever the reason, it is a fact that matte-slugs will frequently carry from  $\frac{3}{4}$  to  $1\frac{1}{2}$  per cent. lead, while the corresponding ore slags made with a lower percentage of fuel will contain only 0.6 to 0.9 per cent. In order to improve reduction, it is quite common to add light scrap iron to the charge, which reduces somewhat the lead carried away, though it necessarily increases the tendency to form speiss. This, indeed, is quite commonly made in concentration smelting, especially when the proportion of lead in the roasted ore matte is low. Sometimes it is a true speiss of the general formula  $\text{Fe}_5\text{As}_3$ , more often, however, the iron is in excess of that proportion, especially when scrap iron has been added to the charge. When scrap iron has not been used, occasionally a kind of speiss will be formed which is not an iron arsenide, but a complex rich in lead and copper, containing up to 30 per cent. of the former and 45 per cent. of the latter, being, moreover, rich in silver, which iron speisses never are. The Freiberg speiss, No. 10 in Table XXVII., seems to be of this class, as is also No. 11 in the same Table.

At *Murray* (Utah)\* the roasted matte from eight furnaces is smelted in one filled-crucible furnace yielding matte of 40 to 50 per cent. copper, which is sent to the Garfield plant. The slags are said to contain only 0.6 per cent. Pb and 0.15 per cent. Cu.

At the *Globe* † works the ore-matte with 10 to 12 per cent. Cu, 12 to 15 per cent. Pb, 40 ozs. Ag, and 0.05 oz. Au per ton is crushed to  $\frac{1}{4}$  inch mesh, roasted in hand-rabbed reverberatories, and then smelted to a second matte with 45 to 52 per cent. Cu.

Such lead as is reduced from the roasted matte is always very rich in silver ; it is important, therefore, to have the bottom of the crucible as impervious as possible in order to minimise absorption, and to handle what is tapped out carefully to diminish losses. When matte and lead are tapped out together a catch-pot with a taphole near the bottom is frequently used under the taphole and from this the matte overflows into other pots. The lead may then be tapped out separately into moulds after the matte in the first pot has begun to solidify, or the catch-pot may be removed from the furnace and a hook inserted in the matte before it solidifies, so that after solidification the cake may be lifted off and the lead poured into moulds.

The further treatment of the enriched copper mattes obtained from lead works does not differ from that of ordinary argentiferous mattes obtained from copper ores direct. Their treatment will, therefore, be most appropriately dealt with in the volume in this series on the *Metallurgy of Silver*.

**Nickel and Cobalt in Matte.**—The occurrence of nickel and cobalt in lead matte is unusual and worthy of special mention. At *Mine La Motte* (Mo.) ‡ ores containing Ni and Co (about 2 of the former to 1 of the latter), the roasting and smelting of which has been already referred to, yield a first matte containing 3 to  $3\frac{1}{2}$  per cent. Ni and Co, 0.5 to 1 per cent. Cu, and 20 to 25 per

\* *Min. Ind.*, vol. xv., p. 667.

† *Min. Ind.*, vol. xiv., p. 398.

‡ Neill, *Trans. A.I.M.E.*, vol. xiii., p. 634.



TABLE XXVII.—ANALYSES OF SPEISSES.

Reference,	1	2	3	4	5	6	7	8	Mayhui, Mexico.	Friedberg, Saxony.	Silver Refinery.
Formula,											
Ag,	0.0085	0.014	0.0270	0.037	0.020	0.017	0.0109	0.049	0.134	0.077	1.06
Au,	tr.	tr.	0.0013	tr.	tr.	tr.	tr.	tr.	0.0007	.001*	0.026
Pb,	1.4935	0.690	2.1800	1.752	3.245	1.9	0.50	3.50	4.6	11.2	29.1
Cu,	0.3628	5.060	1.0600	1.956	0.409	3.0	0.63	7.8	7.8	24.3	31.0
Fe,	60.5780	59.420	57.0200	61.330	56.700	55.0	63.80	71.60	43.3	17.8	11.4
Mn,	..	..	..	..	..	..	..	..	..	0.6	..
Ni and Co,	0.0876	tr.	..	2.056 0.194	0.783	5.1	..	..	..	11.3	..
Zn,	tr.	tr.	0.0700	..	..	2.5	1.75	..	1.3	0.8	..
Mo,	0.2110	tr.	2.3100	..	..	..	..	..	..	..	..
Sn,	..	..	..	..	..	..	..	..	..	..	..
Ti,	..	..	..	..	..	..	..	..	..	..	..
Ca,	..	..	..	..	..	..	..	..	..	..	..
Sb,	..	..	..	..	0.535	..	..	..	0.64	6.5†	..
As,	tr.	tr.	0.1300	2.450	1.608	5.9	..	..	35.00	23.4	..
S,	31.4725	31.170	32.9500	18.750	20.747	23.6	20.75	15.60	4.4	3.5	..
Al <sub>2</sub> O <sub>3</sub> ,	5.8191	2.800	3.3400	9.600	10.000	3.0	4.61	5.5	..	..	..
CaO,	..	..	..	..	..	..	4.96	..	..	..	..
MgO,	..	..	0.3500	..	..	..	..	..	..	..	..
SiO <sub>2</sub> , and insoluble,	..	..	0.2300	..	..	..	0.48	..	1.0	..	0.70
Ag oz per ton,	2.8	4.6	8.8	12	6.5	98.45	97.48	16	44	25	310
Au "	..	..	0.42	..	..	..	0.016	0.089	0.22	0.33	7.8

References.—1. Emmons, "Geol. and Min. Ind. of Leadville," *Mon. XII. U.S. Geol. Surv.*, p. 720. 2. Dewey, *Bull. No. 42 U.S. Nat. Museum*, p. 52. 3. Curtis, "Silver-Lead Deposits of Eureka," *Mon. VII. U.S. Geol. Surv.*, p. 160. 4. Balling, *Berg. u. H. Zeitung*, 1867, p. 419. 5. *Ibid.* 6. *Annales des Mines de Belgique*, 1901, p. 237. 7. *Private notes*, 1904. 8. Hahn, *Trans. Inst. Min. Met.*, vol. vii., p. 271. 9. *Private notes*, 1902. 10. *Glückauf*, 1905, xli., pp. 6 and 1165; also Summary in *Min. Ind.*, vol. xiv., p. 414. 11. *Min. Ind.*, vol. vii., 1899, p. 468.

\* Also 0.007 per cent. Pt.

† Including some Sn.

cent. Pb. This is roasted in reverberatories and used as iron flux in the ore furnaces. According to Neill, it is necessary to leave 5 to 6 per cent. of sulphur in order to prevent slagging away Ni and Co. The second matte containing 5 to 6 per cent. Ni and Co, 1 to 2 per cent. Cu, and 20 to 30 per cent. Pb is roasted and smelted in a similar way, except that in this case 7 to 9 per cent. sulphur is required to protect the nickel and cobalt.

Further concentration of the matte cannot be effected even in a reverberatory, as the lead present appears to drive the Ni and Co into the slag. By taking advantage, however, of the well-known affinity of nickel for arsenic and smelting with mispickel or poor iron speiss, Neill found that a rich speiss of nickel and cobalt could be made, while the reduced lead carried most of the precious metals. The furnace used was 30 inches wide at the tuyeres and 6 feet 6 inches high from tuyeres to charging door, it had a crucible 18 inches deep with syphon tap for the lead. The slag in this case contained only 0.16 to 0.32 per cent. Ni and Co, and 0.6 to 0.8 per cent. Pb; the impoverished matte, 3.25 per cent. Ni and Co, 8 per cent. Pb, and 7 per cent. Cu; while the rich nickel cobalt speiss contained 22.5 per cent. Ni and Co, 6.4 per cent. Pb, and 4.25 per cent. Cu: the silver and gold contents of the reduced lead paid for the whole cost of the operation

### 3. SPEISS.

**Composition.**—The composition of lead speisses is given in Table XXVII., from which it will be seen that they are chiefly arsenides of iron containing more or less sulphur, copper, nickel, cobalt, and lead, the last of which exists, however, chiefly as shots included mechanically. Guyard found also as much as 10 per cent. of metallic iron in the form of shots in speisses from Leadville furnaces, and the author can confirm from his own experience the existence of metallic iron in some speisses.

Arsenides of iron corresponding to the formulæ  $\text{Fe}_2\text{As}_2$ ,  $\text{Fe}_3\text{As}_2$ , and  $\text{Fe}_4\text{As}_2$  can be prepared artificially; they are fusible at a high temperature, yielding white, fine-grained, brittle masses, and speisses of this composition are occasionally produced in lead furnaces, as, for example, No. 9 in Table XXVII. The arsenides of formulæ  $\text{Fe}_5\text{As}_2$  and  $\text{Fe}_6\text{As}_2$  are those generally formed in furnaces smelting not only lead but (under certain circumstances) copper ore also. They are much more fusible than the arsenides lower in iron, and coarsely crystallise in large plates resembling "spiegel-eisen." Speisses still richer in iron corresponding with the formulæ  $\text{Fe}_8\text{As}_2$ ,  $\text{Fe}_9\text{As}_2$ , and  $\text{Fe}_{10}\text{As}_2$  are occasionally formed, but, like those under  $\text{Fe}_5\text{As}_2$ , they are again fine grained, tough, and comparatively infusible, more resembling "sows" than true speisses, and doubtless in every case they contain more or less free iron, as, for instance, No. 8 of Table XXVII. It is noteworthy that whereas speisses always carry much less silver and lead than mattes produced at the same time (fine-grained varieties, however, such as Nos. 8 and 9 of the Table, always containing more than those which are coarsely crystalline) they carry much more gold. This is probably to be accounted for by the presence in them of free iron, which, as is now well known, dissolves gold and has a strong affinity for it.

According to Friedrich,\* there is only one definite compound of iron and arsenic, the composition of which is 40.1 per cent. As and 59.9 per cent. Fe, corresponding with the formula  $\text{Fe}_2\text{As}$  (or  $\text{Fe}_4\text{As}_2$ ), its melting point is  $900^\circ \text{C}$ . Alloys can be formed containing up to 56 per cent. As (about  $\text{Fe}_2\text{As}$ ), but they are not definite compounds. There is an eutectic of  $\text{Fe} - \text{Fe}_2\text{As}$ , having the composition 70 per cent. Fe and 30 per cent. As, the melting point of which is  $830^\circ \text{C}$ . (corresponding with a formula  $\text{Fe}_6\text{As}_2$ ), above which point all other combinations of iron and arsenic contain increasing amounts of free iron.

In modern practice using high furnaces, plenty of blast, slags fairly high in silica, and a good matte-fall, speisses are not nearly so often made as in the practice of twenty years ago employing low furnaces with ferruginous slags low in silica, over-reduction, and frequent, or, indeed, often almost continuous, additions of scrap iron to the charge. When the quantity of arsenic present is high a certain amount of speiss is necessarily formed, but when low the arsenic can be eliminated partly in the flue-dust and partly in the form of matte, whence again it is partly got rid of in the process of roasting prior to concentration, the remainder often forming speiss in the concentration-smelting, as has been seen.

Not only do irony slags encourage the formation of speiss, but, owing to their low melting point, they greatly increase the difficulty of keeping it fluid, and increase the tendency for it to chill, and so form a hearth accretion; with arsenical ores, therefore, it is always best to aim at a high-silica lime slag with a rather high formation temperature, keeping a heavy matte fall with a low fuel consumption and, consequently, low reduction. Should a little speiss begin to form it can often be checked by cutting down the lime flux, which forces the excess silica to combine with iron.

**Precious Metal Contents of Speiss.**—Some figures as to the precious metal contents of speisses are given in Table XXVII. The silver contents of speiss are never high unless the lead is also high, and always bear a direct proportion to the percentages of lead and copper present, and to the richness of the work-lead made at the same time. The gold contents of speiss, however, are often surprisingly high, and it is noteworthy that when speiss is made it is, although poorer in silver, invariably richer in gold than the matte made at the same time. Thus, Iles mentions† that whereas over a series of years his matte averaged 77 ozs. silver and only 0.08 oz. gold per short ton, the average of the speiss made was 24 ozs. silver and  $\frac{1}{2}$  oz. gold per ton. At a well-known works in Mexico,‡ where the matte-fall and speiss fall are about equal, each being a little over 2 per cent. of the weight of the charge, while the matte carries 76 ozs. silver and 0.093 oz. gold per ton, the speiss with only 39 ozs. silver carries an average of 0.1972 oz. per ton, and this in spite of the fact that the lead content of the speiss is less than one-half of that in the matte. We may in fact say that, as a rule, in blast-furnace lead smelting when matte and speiss are made, whilst the matte will not contain in proportion to its silver contents nearly as much gold as does the bullion made at the same time, any speiss produced will show a higher ratio of gold to silver contents than the work-lead itself. The figures from the *Globe Works* reported by Iles may be quoted in illustration of this point as follows:

\* *Metallurgie*, 1907, vol. iv., p. 129.

† *Metallurgy of Lead*, p. 132.

‡ *Private Notes and Communication*, 1904.

	Ore.	Work-Lead.	Matte.	Speiss.
Assay for silver, oz. per ton.	41·00	266	77·0	24·7
"    gold,    "	0·438	3·49	0·08	0·48
Ratio Ag : Au :: 100 :	1·0683	1·312	0·104	1·94

The high efficiency of speiss as a gold-collector is doubtless to be accounted for in part by the free iron in it, that metal having a powerful affinity for gold, for it is probable, as we have seen, that a good part of the iron present in ordinary speisses exists free and simply dissolved in the chemical combination of iron and arsenic, and certain that metallic iron has a powerful affinity for gold.\* Bismuth and antimony, too, are often found in speiss, and both of these elements have a strong affinity for gold, if we may judge from their frequent close association with it in nature. The fact that nickel and cobalt, if present in the ores, are always concentrated in any speiss made may also help to explain the presence of gold in speiss, inasmuch as it has been shown recently that gold is sometimes closely associated with nickel and cobalt.†

**Treatment.**—The speisses produced in lead works ordinarily contain 2 to 4 per cent. Pb, with from 5 to 16 ozs. silver, and 1 to 3 dwts. gold per ton. Their treatment, so as to extract all their silver, gold, and copper contents, offers some difficulties. At some works, when only a small quantity is occasionally produced, it is simply returned to the ore furnace. At most works, however, it is roasted in a reverberatory or in heaps of about 50 tons (badly roasted portions being returned to a new heap) and used as iron flux alone or together with roasted matte. By this means the greater part of the lead, gold, and silver contents of the speiss are taken up by the work-lead, the copper going into the second matte, while the second speiss produced is poorer than the original, besides being much less in quantity, and simply goes to a fresh heap. Where the speiss is very low-grade, as at *Laurium* (No. 7 of Table XXVII.), it is simply allowed to accumulate in heaps separate from the slag, on the chance that some method of utilising it may be found at a future date.

When the ores smelted contain nickel or cobalt, and, at the same time, arsenic (as not infrequently happens), these metals form a speiss instead of going into the matte, and are much more perfectly concentrated into the speiss whenever any is made, the affinity of both metals for arsenic being greater than for sulphur. By roasting and re-smelting these metals become still further concentrated.

At Freiberg in 1860 a mixed matte and speiss containing nickel and cobalt with 163 ozs. silver was smelted without roasting in small reverberatory furnaces together with quartz and heavy spar, the effect being equivalent to roasting off the sulphur, according to Schweder's reaction. After skimming off the slag, metallic lead was added in equal quantity to the original speiss, the whole well rabbled, and the lead tapped off again with nearly all the silver contents of the original speiss; the cleaned speiss contained barely 7 ozs. silver per ton, but was richer in nickel and cobalt than the original.

A modification of this process was introduced at the Eureka Consolidated Works, Nevada, by Davies,‡ who employed an air blast for mixing the lead

\* *v. Fulton and Knudsen, Trans. A.I.M.E., vol. xxxv., p. 336; also Bolles, ibid., vol. xxxv., p. 674.*

† Author, *Trans. I.M.M., vol. viii., p. 303.*

‡ *E. and M. J., June 20, 1888.*

and speiss, and dispensed with the preliminary oxidising fusion. A small cylindrical converter, lined with firebrick, was employed, holding at a charge 800 lbs. of cupola-melted speiss; to this was added from 20 to 25 per cent. of melted lead, and then a blast of air at about 17 ozs. pressure was passed through the mixture for several minutes by means of a  $\frac{1}{4}$ -inch pipe at the bottom of the converter. Some arsenic was burnt off, and some iron slagged, combining with the siliceous lining material; while most of the silver and gold contents were taken up by the lead. The converter was revolved by hand, and emptied into a slag pot having a small taphole in the bottom, through which the still liquid lead was tapped after the speiss had set. The lead now contained 30 to 40 ozs. silver, and the speiss was poor enough to throw away. According to the results given by Davies, in the case of a speiss containing 9 to 10 ozs. silver, 85 per cent. of this metal and 89 per cent. of the gold were extracted, with a loss of 5 to 8 per cent. of the lead employed. The process was, however, costly to work, as well as inconvenient, owing to the fumes of lead, and in most cases a roasting process would be decidedly preferable.

More modern practice at Freiberg is to concentrate dead-roasted speiss separately in a small Pils furnace together with pyrites, galena, matte-slag, litharge poor in silver, old cupel hearths, and some fluorspar as flux for the latter. The product is a second speiss poor in silver, but rich in nickel and cobalt, which is subjected to a second similar concentration, and then sent to the smelting works to be worked up into metallic nickel, cobalt glass, and a lead which goes to the liquation furnace. Provided the iron in the speiss is not under 25 per cent., a matte can be separated carrying most of the silver, lead, and copper, but if the iron is lower than 25 per cent. it becomes difficult to separate the sulphides from the arsenides, as they appear to dissolve each other.

More recently still, the speiss at Freiberg has altered in composition, carrying more copper and less iron, as shown by analysis 10, Table XXVII.\* When this speiss is smelted as described, with galena and matte-slag, most of the Au and part of the Ag and Cu are collected in the resulting matte, but the resulting speiss still retains 0.05 per cent. Ag (16 ozs. per ton), as well as 20 per cent. Cu, and its nickel contents cannot be raised above 16 per cent., while the reduced lead does not satisfactorily absorb the Sb and Pt.

A wet process was tried, fusing with sodium bisulphate and leaching with water. An extraction was obtained of 85 to 97 per cent. of the nickel and copper, but the following difficulties were experienced:—

1. The Pt partly went into solution.
2. The precipitation of the copper from the solution was difficult;  $\text{H}_2\text{S}$  gives a bulky, slimy, and easily oxidised precipitate, and electrolysis is too expensive.
3. The precipitation of Ni was not satisfactory.

Wet processes were, therefore, abandoned, and the latest method is to crush to 5 mm., roast dead, mix with 20 per cent. by weight of pyrites, and smelt. Heavy white fumes of  $\text{Sb}_2\text{O}_4$  and  $\text{As}_2\text{O}_3$  are given off and partially reduced by the pyrites. The concentrated speiss produced in 1904 contained Fe 24.8, Ni and Co 19.6, Pb 7.1, Cu 10.9, As and Sb 27.0, Ag 0.06, Au 0.0002, Pt 0.014.

\* *Glückauf*, 1905, xli., pp. 6 and 1165; abstracted in *Min. Ind.*, vol. xiv., p. 414.

At *Oker* \* (Harz) two classes of speiss are made—namely, the so-called *poor* speiss from the ore furnaces and *rich* from the concentration of roasted matte. Both classes are very rich in copper and in lead, and are indeed rather base antimonial alloys of copper and lead than true speisses, the principal constituent of which is an arsenide of iron. They are treated by oxidising-smelting in reverberatory furnaces with surface blast, the products being a lead-iron-antimony slag, which is returned to the ore furnace, and impure argentiferous copper, which is treated by a wet method. Details of the smelting of the speisses, as well as the further treatment of the copper, will be found in the volume on the *Metallurgy of Silver*.

At the *Germania Works* (Utah)† speiss is roasted (after crushing) in Brückner cylinders, together with the ordinary sulphide ores, in any proportion up to 20 per cent., by which means nearly all of the arsenic is expelled as  $\text{As}_2\text{S}_3$ , which oxidises at once to  $\text{As}_2\text{O}_3$  and  $\text{SO}_2$ . The mixture of roasted ore and speiss then goes to the ore furnaces, and requires much less flux than ore alone, owing to the ferric oxide into which the roasted speiss has been chiefly converted. This may be considered the best mode of dealing with speiss, when it does not contain sufficient nickel and cobalt to justify the adoption of the Freiberg separate treatment, and yet carries lead and silver or gold enough to warrant some attempt to extract these metals.

#### 4. SLAG.

**Composition.** The constitution of lead slags has been fully discussed in Chapter vii., and their handling outside the furnace and separation from matte in the last chapter. Some analyses of lead slags have been collected in Table XXVIII., and others are given in Table XXXII. (Chap. xi.), which, however, are not individual samples, but averages.

It is difficult to tabulate the analyses of some lead slags, on account of the doubt as to the mode of combination of Ba and of S. In slags comparatively low in zinc there can be no doubt that the S present is practically all combined with Fe, Pb, Cu, &c., in the condition of matte so finely disseminated as to be virtually in a state of solution, as shown by Mrázek.‡ This matte can be separated by fusion with borax, and contains practically all the valuable materials present in the slag. In non-barytic zinciferous slags most of the S may be considered as combined with Zn and Fe, though occasionally CaS may be present in slags from low furnaces. When much barium is present, part of it is always found as sulphide, BaS, which burns with the characteristic green flame when the slag is tapped; the remainder, in modern high pressure furnaces, usually exists as silicate,  $\text{BaSiO}_3$ , though in low furnaces, where reduction is not so powerful, most of it enters the slag unaltered as  $\text{BaSO}_4$ . Thus the slag produced at the *Juliusshütte bei Goslar*, No. 2 of Table XXVIII., probably contained its Ba mostly as  $\text{BaSO}_4$ , though partly, perhaps, also as BaS; this slag certainly contained also some Zn as ZnS, as no doubt did the slags from Freiberg and Clausthal, and possibly others.

\* *Glückauf*, 1905, xli., abstracted in *Min. Ind.*, vol. xiv., p. 414.

† *Private Notes*, 1897.

‡ *B. u. H. Jahrbuch*, 1866, vol. xvi., p. 392; quoted by Percy, *Met. of Lead*, pp. 407-409.

TABLE XXVIII.—ANALYSES OF AVERAGE LEAD BLAST-FURNACE SLAGS.

Reference,	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Muldener Hütte, Freiberg.	23.95 4.45 44.41 0.92 14.81 4.75 0.54 und.	16.90 6.31 35.05 .. 19.64 6.05 0.60 und.	31.09 6.98 40.64 0.86 8.01 4.39 0.05 0.58 0.35	35.79 4.81 29.50 2.10 6.00 16.50 1.58 .. ..	27.96 2.33 35.81 4.74 6.32 13.90 3.18 .. ..	26.70 3.10 32.95 .. 12.88 19.71 1.80 .. ..	25.20 11.38 29.88 2.88 12.34 17.33 tr. .. ..	31.94 2.57 35.42 1.52 1.91 20.42 0.78 0.45 0.34	23.67 1.64 58.32 0.23 4.44 4.78 1.27 und. }	35.70 11.91 12.09 24.04 und. 12.02 2.80 .. ..	35.30 5.56 35.90 .. .. 13.90 9.27 .. ..	31.37 6.10 28.57 5.86 6.17 14.22 2.87 .. ..	35.00 3.00 31.00 .. 5.00 23.00 .. ..	26.40 1.9 31.9 5.5 8.2 19.2 .. ..	40.4 9.8 13.3 13.2 3.4 13.4 1.4 ..	25.0 6.0 33.0 6.0 13.0 12.0 .. ..	33.35 5.03 35.07 .. 6.23 16.80 .. ..
Australian S. & R. Co., Dapto, 1903.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
B. H. P. Co., Port Pirie, 1906.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Broken Hill Prop. Co., Broken Hill, 1896.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Mapimi, Mexico.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Monterey, Mexico.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Globe, Denver, Colo.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Mine La Motte, Mo.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Tombstone, Arizona.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Eureka, Nevada.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Mazarron, Spain.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Laurium, Greece.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Pertusola, Italy.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Overpelt, Belgium.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Przibram, Bohemia.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Clausthal, Upper Harz.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Julfushütte bei Goslar, Lower Harz.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Ag oz. per ton,	101.62	98.73	101.002	101.0625	97.92	99.249	100.591	99.221	101.2318	100.206	100.45	101.42	98.70	97.31	94.9	99.5	97.69
Au "	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..

References.—1 and 2. Schnabel, *Handbuch der Metallhüttenkunde*, vol. i., p. 348. 3. *Eng. and Min. Journ.*, Oct. 21, 1893. 4. Cap, *Oesterr. Zeitschr. f. Berg- u. Hüttenwesen*, 1900, p. 231. 5. *Firket, Ann. des Mines de Belgique*, 1901, p. 237, abstr. in *Min. Ind.*, 1901, vol. ix., p. 427. 6. *Berg- u. Hütten. Zeitung*, 1899, p. 189. 7. *Private notes*, 1908. 8. Sanchez Massia, *Metallurgia del Plomo*, Madrid, 1893. 9. Curtis, *Mon. VII. U.S. Geol. Surv.*, 1889, p. 160. 10. Chureh, *Trans. A.I.M.E.*, vol. xxiv., p. 561. 11. Kemp, *Sch. of M. Quarterly*, vol. ix., p. 218. 12. *Iles, Metallurgy of Lead*, p. 131. 13 and 14. *Private notes*, 1902. 15. *Private notes*, 1897. 16. Delprat, *Trans. Aust. I.M.E.*, vol. xii., pp. 1-29. 17. *Priv. communication*, 1904.

Note.—Some of the above analyses have been re-calculated from the original data.

Potash and soda are seldom determined in commercial slag analyses, yet they are often present in small quantity, as shown by Hles \* and Canby,† the latter, indeed, suggests that the abnormally high tonnage capacity reached by some furnaces handling certain ores may be partly due to the presence of alkalies.

It was pointed out by Pearce ‡ that certain reverberatory slags made at Argo contained considerable proportions of magnetic material, which he assumed to be magnetite, and to be formed by the oxidising influence of the furnace gases upon the slag after the reducing effect of S and SO<sub>2</sub> had terminated. E. Keller has since shown § that much Fe<sub>3</sub>O<sub>4</sub> (up to 18 per cent.) is frequently found in the slags produced by smelting roasted mattes in reverberatory furnaces, and smaller amounts are invariably present in matte slags produced in blast furnaces, the atmosphere of which is not sufficiently reducing to destroy this substance before it is taken up by the molten products. When the matte is of low grade and has a specific gravity not over 5·2 (the specific gravity of magnetite), the latter substance sinks into it in preference to going into the slag. When, however, a high grade copper or lead matte is being made, the specific gravity of which is over 5·3 or about that, as in concentration smelting, the magnetite cannot enter the matte, and is carried away by the slag.

**Metallic Contents of Slags.**—The chief requisites for obtaining a clean slag are :—

1. The correct composition of the slag.
2. The bullion-fall, which, under exceptional circumstances, may be as low as 7 per cent., but should preferably reach 10 per cent. at least.
3. The matte-fall, which should be not less than 4 or 5 per cent., and preferably 7 to 12 per cent., providing the matte is a good irony matte with a fair amount of copper and little zinc.
4. Proper means of separating matte from slag.

Besides these the richness of the ore charge is an important factor in determining the absolute losses of metals, particularly of silver. Thus, with a favourable ore mixture yielding 14 per cent. of work-lead, containing only 200 to 300 ozs. of silver per ton, slags can be sometimes made as low as  $\frac{1}{2}$  per cent. Pb and  $\frac{1}{2}$  oz. Ag per ton, but on an average 1 per cent. Pb and  $\frac{3}{4}$  oz. Ag may be considered good work. With a production of 11 per cent. of work-lead running up to 400 ozs. of silver per ton the slags, although occasionally almost as clean, will average not under  $1\frac{1}{4}$  per cent. lead and  $1\frac{1}{4}$  to  $1\frac{1}{2}$  ozs. of silver.

The losses of valuable metals in the slag take place in the following ways :— (1) Through fine visible shots of matte and occasionally still more minute ones of work-lead; (2) through incomplete reduction of metallic oxides; and (3) through an almost infinitely fine dissemination of metallic and sulphide particles which are held by the slag, either in solution or in such intimate suspension as practically to amount to a solution. The loss in the first way is usually the most important, and can be almost completely eliminated by careful settling outside the furnace, as has been already shown. The other two sources of loss can only be diminished by the use of an independent reverberatory settler in which a reducing flame is kept up, or by precipita-

\* *E. and M. J.*, June 3, 1889, p. 650.

† *Trans. A.I.M.E.*, vol. xviii., p. 10.

‡ *Ibid.*, July 1, 1889, p. 3.

§ *E. and M. J.*, Nov. 16, 1895.



tion by scrap iron or pyrites in a suitable form of settler. Most of the silver contained in a slag may be considered as existing in the condition of suspended matte, as is also most of the lead contained in clean slags from modern high pressure and high column furnaces. The slags from lower furnaces are, however, always richer in lead, especially when somewhat siliceous (as at Broken Hill), and in such cases part, if not most, of the lead probably exists as silicate owing to imperfect reduction. The question of slag losses being further dealt with in the Chapter on *Costs and Losses*, it will be sufficient to add here that the precious metal losses appear to be entirely mechanical, while that of lead is evidently in some cases through the existence of undecomposed chemical combinations in the slag.

**Removal of Slag and its Utilisation.**—The method of removal by granulation has been already referred to in the Chapter on *Plant*. When granulation is not adopted, the slag must be either cast in moulds or blocks suitable for loading upon railway trucks, or it must be formed into slag dumps, which are afterwards broken down and loaded up by ordinary methods, a process that is, however, somewhat costly. Methods of utilising slag may be classified as follows:—(1) Simple utilisation as ballast or as aggregate for concrete; (2) utilisation in blocks or bricks as building material; (3) utilisation of the heat in molten slag; and (4) chemical methods by which certain of the ingredients in the slag are extracted and rendered valuable.

**Use of Slag as Ballast.**—Granulated slag, although largely used, is not of great value as ballast, since it does not pack well, but it will serve instead of sand or gravel for concrete and for mortar-making, provided mortar-mills are used.

A common mode of fitting slag for ballast at old-established works limited as to storage room is to pour it into iron moulds, which are (sometimes after spraying while hot to fissure the blocks) broken up with sledges and put through stonebreakers. Slag quarried from old slag dumps made by means of hand pots frequently breaks up so as to be in ideal condition for use as ballast with little or no crushing.

In one or two cases slag has been cast in rows of moulds slung upon continuous chains like a pan conveyor, and passing, at some point in their course, under or through water sprays, or even through a shallow tank of water, in order to cool them and diminish the wear and tear; but these methods do not appear to have come into use to any extent at lead-smelting works, though employed at some large copper and iron plants.

An ingenious method of automatically handling slag and delivering it in suitable condition for use as ballast is that devised by Page of the Omaha and Grant Works, Omaha.\* About 75 feet from the furnace building is a circular pit, 35 feet diameter, at the edge of which revolves an annular disc composed of slightly dished cast-iron plates, 2 feet wide and  $1\frac{3}{4}$  inches thick, which runs by means of rollers on a circular cast-iron submerged track resting on pillars all round the pit. A toothed wheel on the circumference of the disc engages with a pinion, by means of which it is revolved at the rate of one turn in five minutes. A constant stream of cold water enters the pit, and the hot water is run away just so fast as to keep the lower side of the cast-iron plates under water. As each slag pot is tapped on to the revolving plates the slag spreads out into a layer  $\frac{1}{4}$  inch to 1 inch thick, and cools

\* Braden, *Trans. A.I.M.E.*, vol. xxvi, p. 51.

quickly, cracking up in so doing, so that at the end of the revolution a fixed plate of  $\frac{1}{2}$ -inch steel against which the layer of slag impinges readily breaks it up into flat irregular pieces from  $\frac{1}{2}$  inch to 4 inches long and wide by  $\frac{1}{4}$  inch to 1 inch thick, and forces these to pile themselves on to a conveyor, which delivers the slag into bins from which the railroad trucks are loaded.

**Use of Slag as Building Material.**—Iron blast-furnace slags have been utilised as building material both by casting them into slag bricks, and (as, for example, at Siegen) by granulating, grinding fine, calcining with limestone, and re-grinding for the manufacture of hydraulic cement. The slags in question, however, consist chiefly of an acid silicate of alumina and lime, and it is doubtful whether lead slags, which are much more basic and contain scarcely any alumina, could, under any circumstances, be successfully utilised for the manufacture of cement, the results of experiments made up to the present being decidedly unfavourable.

**Manufacture of Slag Brick.**—To some extent the same objection is to be noted here, the best slags for slag-brick purposes being those containing upwards of 40 per cent. silica, which are not produced in lead smelting. Lead slags are generally too brittle and too much inclined to fissure on cooling, to make the best kind of building and paving brick, such as that produced at Mansfeld. Still, at most lead-works a few bricks are made for use in building ore-bins, foundations, dust chambers, &c., and there is no doubt that this could be extended with advantage. The manufacture of slag brick from reverberatory copper slag has been described by Dr. Peters,\* and that from blast-furnace slag at Mansfeld,† and in Montana,‡ is well described in two papers in the *School of Mines Quarterly*, to which reference should be made for details. According to the author's experience in making slag brick it is of the utmost importance that the slag should be as cool as possible before casting, and that the bricks should be cooled as slowly as possible by removing the moulds as soon as set and covering them several inches deep in sand. Dusting a little sand into the moulds as they are being filled also assists in the production of a slightly rough surface which is better for holding mortar than a smooth glassy one.

By proper annealing and toughening even paving tiles may be made from ferruginous blast-furnace slags. Thus, at Santiago de Chili § the slag is ladled from the slag pots into cast-iron moulds 6 inches square by 1 inch deep, each provided with a cover. After filling a lot of moulds they are covered and placed on a hearth, which has also a cover for annealing, a gentle heat being kept up so that the cooling operation may be slow. When the slag has cooled to a black heat, the moulds are lifted from the hearth and the tiles thrown into cold water to toughen them. The tiles are very hard and durable, and are worth from £2 to £4 per 1,000.

An entirely distinct class of slag-brick is the slag-lime brick so largely made at many ironworks, particularly in Germany. The slag is granulated by a stream of water and then mixed with slaked lime in the proportion of 1 part of quicklime to 6 of granulated slag. The mixture is passed through a set of rolls and then through a mixing machine; the bricks are moulded in a machine which has a hopper filled by a labourer, and which turns out the

\* *Modern Copper Smelting*, 7th Ed., 1895, p. 142.

† Egleston, *S.M.Q.*, vol. xii., pp. 112-117.

§ Braden, *Trans. A.I.M.E.*, vol. xxvi., p. 52.

‡ *Ibid.*, p. 189.

bricks automatically, subjecting each to a great pressure by means of cams. One pair of machines turns out 10,000 bricks per day, requiring 3 H.P. for the mixer and 8 H.P. for the moulder. The bricks are at first tender, and must be stacked carefully and protected from rain; but after six or twelve months' exposure to air they become solid enough for building. At the *Karl Emil Works* (Bohemia)\* during 1895 12,000,000 bricks were made by this method from a slag containing  $\text{SiO}_2$  26 to 27 per cent.,  $\text{Al}_2\text{O}_3$  17.3 to 19.3,  $\text{CaO}$  51.5,  $\text{Fe}_2\text{O}_3$  1.7,  $\text{MgO}$  0.4 to 2.5, and S 1.3 to 1.8 per cent.; in this case the bricks set hard enough for shipment in eight days.

The manufacture of slag-lime bricks has been widely extended during the past few years, but always employing slags from iron blast furnaces, and it seems doubtful whether slags from lead blast furnaces containing, as they do, not only a low percentage of lime and alumina, but quite high proportions of ferrous iron, and sometimes also of sulphur, would give successful results.

**Utilisation of the Heat in Molten Slag.**—Attempts have been made to utilise the waste heat of molten slag for the purpose of heating the blast, and, as is described in the volume in this series on the *Metallurgy of Silver*, so far as matte-smelting is concerned, such attempts not only contain a germ of usefulness, but have already attained a fair measure of success. The use of hot blast in lead smelting, however, is of doubtful advantage, and would probably result in enormously increasing the volatilisation losses, so that the field scarcely seems a promising one for experiments.

Attempts were made many years ago at Broken Hill by Howell and Ashcroft to utilise the heat of molten slag for raising steam. The apparatus consisted of a flat egg-ended steel-plate boiler containing vertical tubes of copper or wrought iron completely surrounded by water; the tubes were tapering, with the larger end downwards, and closed by cast-iron doors so arranged as to be easily opened and closed from one side by a system of lever handles. The boiler shell was fitted with a steam-dome safety valve, and other usual fittings, and sheet-iron bins lined with firebrick were provided at each end of the boiler to receive the "shells" from the pots, so as to help in keeping up the heat in the boiler. Both the "shells" and the cooled slag in the tubes were emptied into an iron truck running on a track beneath the boiler. The taper tubes were filled successively with slag by means of the ordinary slag pots, the first being emptied as soon as the last was filled. Unfortunately, it was found that the shells of slag formed in the tubes were such bad conductors of heat that the actual pressure of steam attained was only 20 to 30 lbs., an amount quite insufficient to be of any practical service under the local conditions then prevailing at Broken Hill.

Quite recently the improvements in low-pressure steam turbines, which, working against a good vacuum, enable steam pressures of as low as 1 lb. to be rendered economically available, have again directed attention to the problem of heat-utilisation from waste slag. In the ingenious device of Vautin the stream of slag from the forehearth, or preferably from a settling reverberatory, is granulated by hot jacket-water in a hopper-shaped vessel, which is sealed as regards the inflow of slag by making the slag-stream flow through a "trap," and as regards the outflow of slag granules by discharging these by means of an ordinary vertical bucket elevator with perforated buckets working in a separate compartment, which communicates with the

\* *E. and M. J.*, Dec. 6, 1896, p. 510.

hopper only at its point, thus forming an effective water-seal. This system appears to be still in the experimental stage, and one obvious drawback is the necessarily heavy wear and tear on the slag-trap, and the difficulty of clearing out or making repairs to this essential part of the apparatus; the power realisable, however, which is estimated to be 1 H.P. per ton of slag per day, would seem to justify a good deal of effort to over-ride such difficulties.

**Chemical Methods.**—At *Langenheim* (Lower Hartz) the lead blast-furnace slags rich in zinc and baryta, the composition of which is referred to in connection with the description of lead smelting at Goslar (Chap. xi., *Examples*), are treated as follows:—After fine grinding in a ball mill the slag is melted with a calculated quantity of  $\text{CaCl}_2$  in a reverberatory furnace at a temperature of about  $1,200^\circ \text{C}$ . The melted mass, when cold, is again pulverised in a ball mill and leached with water, giving a solution of nearly pure barium chloride. The moist residue is treated in lead pans with sulphuric acid of  $50^\circ \text{B}$ , which causes it to heat, turn white, and solidify. The mass is broken up, mixed with  $\text{CaCl}_2$ , pulverised, and heated to  $500^\circ \text{C}$ ., whereby chlorides of Fe and Zn are produced, the former being again decomposed with expulsion of HCl, which is condensed by means of powdered limestone to form  $\text{CaCl}_2$ . Upon leaching,  $\text{ZnCl}_2$  is recovered from the solution, and the insoluble residue, consisting mainly of silica, calcium sulphate, and ferric oxide, is so red as to be saleable as a cheap red paint.

**Working up of Old Slags.**—In many localities are found heaps of old slags resulting from the smelting of lead slags in primitive hearths and other furnaces, and these are being profitably worked up at the present time.

At *Laurium* (Greece) two French companies have been for many years past smelting slags produced in the days of Themistocles and Xenophon, together with old slimes rich in silver from the ancient dressing operations, and lead concentrates from modern dressing works (the largest in Europe), treating the low grade ore bodies left behind in the mines. The supply of old slags on land, however, is now exhausted, and only small quantities are still obtained from the sea by dredging; they contain 10 per cent. lead and 13 ozs. of silver per ton. At many other places in Europe similar work has been done in the past upon old slags, but in all the more accessible countries such accumulations of rich slags have been pretty well worked off, and rich slag heaps can only be found in such countries as Peru, Bolivia, and Burmah.

A still better way of cleaning zinciferous or other foul lead slags is by the use of pyritic copper ores. In a furnace at *Leadville*, 36 inches by 80 inches, one month's campaign described by Keller\* consisted in smelting 3,107 tons of slag with 5·3 ozs. Ag per ton and 402 tons of pyritic copper ores, averaging 10 per cent. Cu and 11 ozs. Ag per ton. The matte produced was separated from the slag in an open settler forehearth, and averaged 20 per cent. Cu and 93·3 ozs. Ag, while the cleaned slag averaged under  $\frac{1}{2}$  oz. Ag and  $\frac{1}{2}$  per cent. Cu. The fuel consumption was 10 per cent., and the total cost per ton of material smelted only 6s. 3d.

#### OTHER FURNACE PRODUCTS.

**Furnace Accretions.**—These may be of two kinds—viz., hearth accretions or “sows” and the far commoner “wall” or “shaft accretions.”

\* *Trans. A.I.M.E.*, vol. xxi., p. 71.

TABLE XXIX.—ANALYSES OF FURNACE ACCRETIONS.

Reference, . . .	Sows.		Wall Accretions.				
	Leadville.	Przibram.	Leadville.	Leadville.	Pueblo.	Globe Works, Denver. Average.	Laurium, Greece.
1	2	3	4	5	6	7	
Pb, . . . .	18·793	20·140	47·491	21·317	1·48	24·06	25·00
PbO, . . . .	..	..	0·405	5·245	..	..	..
Zn, . . . .	tr.	5·417	6·977	..	45·68	18·42	24·40
ZnO, . . . .	..	..	0·300	66·534	..	..	..
Fe, . . . .	72·828	38·875	6·754	..	6·71	13·26	13·05
Fe O, . . . .	..	..	1·100	1·381	..	..	..
Mn, . . . .	0·015	tr.	2·010	..	0·54	1·60	1·35
Cu, . . . .	tr.	1·107	tr.	..	none	0·80	tr.
Ag, . . . .	0·1149	0·160	0·0754	0·0944	0·0188	0·0909	0·0105
Au, . . . .	tr.	tr.	tr.	..	..	0·0060	..
As, . . . .	5·083	2·389	0·039	..	none	..	1·40
As <sub>2</sub> O <sub>3</sub> , . . . .	..	..	..	0·071	..	..	..
Sb, . . . .	tr.	0·735	tr.	..	tr.	..	..
Sb <sub>2</sub> O <sub>3</sub> , . . . .	..	..	..	0·056	..	..	..
Sn, . . . .	..	..	..	..	..	..	..
Mo, . . . .	0·161	..	..	..	..	..	..
Ni and Co, . .	0·045	tr.	..	..	..	..	..
P, . . . .	0·109	..	0·945	..	..	..	..
C combined, .	0·550	..	..	..	..	..	..
C graphitic, .	0·750	..	..	..	..	..	..
SiO <sub>2</sub> , . . . .	0·900	9·300	10·100	1·577	8·13	17·26	8·40
Al <sub>2</sub> O <sub>3</sub> , . . . .	..	..	1·672	0·328	..	..	6·36
CaO, . . . .	..	0·780	5·361	0·200	6·20	3·30	4·95
MgO, . . . .	..	..	4·297	0·159	..	..	..
SO <sub>3</sub> , . . . .	..	..	0·965	0·287	..	..	..
Cl and Br, . .	..	..	..	0·035	..	..	..
S, . . . .	0·650	18·466	8·291	2·725	25·98	10·90	14·57
..	..	..	..	..	..	..	..

References.—1, 3, and 4. Emmons, "Geology and Mining Industry of Leadville," *Mon. XII. U.S. Geol. Surv.*, pp. 723 and 727. 2. Balling, *Berg- u. H. Zeitung*, 1867, p. 419. 5. Dewey, *Bull. No. 42 U.S. Nat. Museum*. 6. Iles, *S.M.Q.*, vol. xviii., pp. 20-21. 7. *Private Notes*, 1904.

"Sows" may form on the hearth of a sump furnace, or above the lead bath in a furnace with crucible which is kept full. They are caused (1) by badly calculated charges; (2) by lack of fuel; and (3) by too slow a rate of smelting. The chief item in their composition is metallic iron, but they also contain lead, sulphur, arsenic, copper, nickel, and occasionally also tungsten, molybdenum, vanadium, phosphorus, titanium, and silicon, besides particles of coke, matte, speiss, &c., mechanically enclosed. When rich in lead and sulphur they are often comparatively rich in precious metals, and are then worth smelting down with siliceous ores on the hearth of a reverberatory furnace. When composed chiefly of metallic iron they are always poor in silver, and are rarely worth working, so that in such cases the common practice is to throw them over the slag dump and bury them up as soon as possible.

It is probable, however, that in some cases notable amounts of gold have been lost in this manner, since metallic iron has, as has been seen, a powerful affinity for gold. In a large works with regular supplies of ores to draw upon and carefully calculated charges they are of rare occurrence; perhaps only one or two in the course of a year. The composition of some "sows" from lead furnaces is given in Table XXIX.

"Wall Accretions" or "Hangings."—These are chiefly composed of volatilised and condensed sulphides of zinc and lead with some oxides of both metals, and often some metallic zinc, cementing fragments of half-smelted charge. The way in which lead and zinc, together with some of their compounds, particularly the sulphides, are volatilised in the smelting zone of the furnace and become condensed in a partially oxidised condition in the upper portions, from just above the jackets more than half-way up to the feed floor, has been already described (Chap. ix.); as has also the method of removing them and smelting them out.

Some analyses of wall accretions are given in Table XXIX.

Furnace Gases.—The gases from lead blast furnaces have not received the same amount of study as have those of pyritic furnaces or those of iron blast furnaces. Hofman gives\* eight analyses of gases from two smelting works in Colorado, which may be repeated here:—

	Works A.				Works B.			
N <sub>2</sub>	74.1	69.4	73.0	68.6	77.6	78.9	79.6	75.5
CO <sub>2</sub>	15.2	17.6	17.4	21.6	18.8	19.6	16.6	18.2
CO	9.7	8.0	5.4	10.8	3.0	1.5	3.6	5.9
O <sub>2</sub>	1.0	5.0	4.2	0.4	0.6	0.0	0.2	0.4

The variation in the ratio of CO to CO<sub>2</sub> is, as will be seen, considerable. It is probable that with high-pressure blast the whole of the oxygen in the air which reaches the centre of the furnace, not counting that proportion which escapes through hugging the inner face of the jackets between them and the charge, is burned to CO<sub>2</sub>, and that a variable proportion of this is retransformed to CO by passing through the heated coke. Some experiments of Austin † show that the proportion of CO is even greater some feet down than at the very top of the charge; thus, whereas at the surface a sample of gas gave CO<sub>2</sub> 20.5 per cent., CO 7 per cent., another sample from 3 feet down, where the charge was a just visible red, gave CO<sub>2</sub> 16.4 per cent., CO 13 per cent. It would perhaps appear that of the CO<sub>2</sub> generated before the tuyeres a large proportion is retransformed into CO in passing through the layer of red hot coke just above the smelting zone; and that a variable proportion of this CO is re-burned to CO<sub>2</sub> much higher up in the furnace by the help of the oxygen which has hugged the sides and so escaped combination lower down. This hypothesis helps to explain the smaller fuel consumption in furnaces with a high charge column.

The subject of flue-dust, its composition, collection, and treatment is dealt with in a separate chapter.

\* *Metallurgy of Lead*, 1906, p. 309.

† *Mining and Scientific Press*, Sept. 12, 1908, p. 364.

## CHAPTER XI.

### BLAST-FURNACE LEAD SMELTING—EXAMPLES.

It will be convenient to consider the chief centres of blast-furnace lead smelting in the following order:—Europe (Germany, Austria-Hungary, Spain, Italy, Greece), Australia, Canada, United States, Mexico, and S. America.

#### GERMANY.

This country and Spain were the original homes of blast-furnace smelting. Labour and fuel being usually cheap as compared with new countries, greater attention is paid to the recovery from the ores treated of as much as possible of their valuable constituents, regardless of the complexity of the operations thus introduced; and occasionally without due regard to the expense and loss caused by too frequent handlings.

**Smelting at Freiberg.**—At this ancient mining centre almost all kinds of ores are treated by a variety of processes which centre round the lead blast furnace, and which result in saving not only the lead, silver, gold, and copper, but also the greater part of the bismuth, arsenic, cobalt, nickel, and a considerable portion of the zinc and sulphur contained in the ores. The nature of these processes and their relationship to each other are well shown in a diagram figured in Prof. Sir W. C. Roberts-Austen's *Introduction*, p. 363, which represents the operations as conducted up to quite recently, except that they have been simplified by bringing up the first matte to 30 per cent. copper matte at one operation; and so dispensing with the third Pilz furnace, except for the purpose of cleaning slag and of smelting speiss.\*

The ores treated may be divided into five principal classes:—

1. Lead ores carrying from 15 to 80 per cent. lead and various amounts of silver, the average being 40 per cent. lead and 110 ozs. silver per ton.
2. Pyrites and copper ores containing from 1 to 10 per cent. copper besides iron, lead, zinc, silver, &c., all as sulphides.
3. Dry ores—that is, silver ores more or less siliceous, but poor in lead.
4. Zinc ores divided into (a) those containing less than 25 per cent. Zn; and (b) those containing upwards of this amount.
5. Arsenical ores containing various amounts of silver, and divided into “mispickel ores” and “mixed arsenical ores.”

Of the above classes 1 and 3 almost entirely go direct to the roasting-furnace mixture beds.

The ores of class 4 containing less than 25 per cent. zinc also go to the roast-furnace beds, while those containing upwards of 25 per cent. are twice

\* v. also Percy, *Metallurgy of Silver*, pp. 544, et seq.; Schnabel, *Handbuch der Metallhüttenkunde*, p. 389, et seq.

roasted, first in kilns and then in reverberatories, and then distilled for zinc by mixing them with charcoal, and heating in retorts in a regenerative gas furnace.

With regard to class 5, it is divided; ores poor in pyrites, which are chiefly arsenical, being roasted in a current of air to produce white arsenic; while the heavy mispickel and pyrites ores, containing sulphur and arsenic in the proportion of about two to one, or thereabouts, are distilled in closed retorts so as to yield realgar.

Such arsenical ores as may contain more than 25 per cent. sulphur and less than 10 per cent. As, together with the whole of the ores of class 2 and a few lots from classes 1 and 3 specially heavy in pyrites, as well as the residues from the realgar works, are roasted together to collect the sulphur for the manufacture of sulphuric acid. The lump ore is roasted in the kilns already referred to in Chapter vi., the fines in Gerstenhöfer and muffle furnaces. It should be noted that the sulphurous acid from the zinc ore roasters and from the arsenic glass works is also utilised in the sulphuric acid works. All these sulphurous gases contain more or less arsenic, which, by conducting the gases through a series of lead-lined dust-chambers before allowing them to enter the lead-chambers, is separated out as  $\text{As}_2\text{O}_3$ ; and this  $\text{As}_2\text{O}_3$  is returned to the arsenic works to be worked up into the finest white arsenic glass.

The ores thus partially roasted in the kilns and Gerstenhöfer furnaces are by no means freed from their sulphur, but no more of it can be utilised with advantage for the manufacture of sulphuric acid; and so the residues go to the mixture beds of the long reverberatory roasters.

These mixture beds, therefore, contain (1) raw galenas carrying 26 to 79 per cent. lead, and 3 to 150 ozs. silver per ton; (2) raw siliceous lead and silver ores; (3) residues from the zinc works; (4) residues from the arsenic works; and (5) once roasted pyrites and coppery ores of all descriptions. Their average composition is—lead, 20 to 30 per cent.; zinc, under 10 per cent.; silica, 20 to 25 per cent.; sulphur, 20 per cent.; and silver, 30 to 50 ozs. per ton. The roasting is incomplete, 3 to 5 per cent. of sulphur being always left in the ore in order to collect the copper in a matte; at the end of the roasting proper the charge is “sintered” down in the sump at the firebridge end of the hearth. The furnace has been already figured and described in Chapter v. (p. 83). The flue-dust from these reverberatory roasters is rich in arsenic, and is treated in the arsenic works.

The sinter-roasted ore is smelted in the Pilz furnace already figured and described in Chapter viii. (p. 173). The charge is composed of roasted ore mixed with about an equal weight of slag from the same furnace, which is used in order to clean the slags and also to assist in carrying away zinc, by which means campaigns are obtained of three or four years' duration. The daily capacity of each furnace is 30 to 35 tons of ore besides the slag, with a consumption of  $5\frac{1}{2}$  tons of coke, equal to 16 per cent. of the ore or  $8\frac{1}{2}$  per cent. on the whole charge; the number of men on each shift of twelve hours is usually five—namely, one feeder and one wheeler on the charging floor; and one smelter, one slag runner, and one odd man on the dump or lower floor. The furnace is worked as a sump furnace, lead and matte being tapped together (and speiss also when produced) into a movable exterior crucible or receiver on wheels, in which the various products separate into layers.

The work-lead produced carries from 200 up to 300 ozs. silver per ton and goes to the liquation and desilverising works, where not only are its gold and



silver contents extracted, but also traces of antimony, bismuth, and tin are recovered commercially. The slags formerly produced were rather more acid than monosilicates; with the increase of zinc contained in the ore, however, it has become necessary to run more basic slags, and those now made (v. Table XXVIII.) are slightly more basic than monosilicates. They contain from 2 up to (occasionally) as much as 6 per cent. Pb, about  $\frac{1}{2}$  per cent. Cu, 9 to 14 per cent. ZnO, and from 1 to 2 ozs. silver per ton. The speiss produced in ore smelting is set aside to be worked up with that produced in subsequent operations. The matte produced contains 15 to 25 per cent. Pb, 8 to 14 per cent. Cu, and 45 to 65 ozs. silver per ton. It is roasted first in kilns (the roast gases going to the sulphuric acid works) and then in Wellner stalls (v. Chap. v.), being afterwards smelted together with the ore-slag, unless containing upwards of 12 per cent. Cu.

**Slag Smelting.**—The roasted matte still containing 5 to 8 per cent. of sulphur is smelted together with from one and a-half to twice its own weight of foul slag from the ore furnace, and a small quantity of lead skimmings and drosses from the tap pits, and residues from the liquation hearths. A charge of 300 kilos. ore-slag, 200 kilos. roasted matte, and 4 kilos. drosses is smelted in a furnace, precisely like the ore furnace, at the rate of 60 tons of mixture per twenty-four hours, with a coke consumption of only 35 kilos. per charge, or, say, 7 per cent. The products are:—

Second work-lead with from 130 to 200 ozs. silver per ton, which goes to the desilverisation works.

Second matte, "copper-lead-matte," or "slag-matte," with 25 to 30 per cent. copper, 5 to 8 per cent. Pb, and 25 to 50 ozs. silver per ton. This is brought up to "pimple metal" of 80 per cent. copper by one or two roastings and fusions; the further treatment of this material is described in the *Metalurgy of Silver*.

The slags produced in slag-smelting contain about 29 per cent. SiO<sub>2</sub>, 37 per cent. FeO, 9 to 12 per cent. ZnO; they carry usually about  $1\frac{1}{2}$  per cent. Pb by wet assay, and  $\frac{1}{2}$  oz. silver per ton, and are clean enough to throw away. Occasionally, however, portions are richer, when they are re-smelted in a smaller Pilz furnace with the addition of about 1 per cent. of lead drosses. In this case the capacity of the furnace is 36 tons with a coke consumption of 11 per cent.

The speiss produced in slag-smelting is generally a good deal more than in ore-smelting. Both are laid aside together till sufficient has been collected for a campaign. It is then roasted and smelted in the small Pilz furnace, together with a large excess of foul ore-slag and smaller quantities of liquation residues, and cupel bottoms, with fluorspar as flux for the latter. An impure work-lead is produced which carries most of the silver in the charge; a small quantity of matte which is added to the first or second matte; according to its richness in copper; and a second speiss, which is worked up as described in Chapter x., after which it goes to the smalt works, where its cobalt and nickel contents are recovered in different forms, copper and lead returning to the matte furnace.

The methods of flue-dust collection employed at Freiberg are described in the next chapter; the total amount collected in 1893 was 9·7 per cent. of the weight of ore smelted, and this was treated by bricking and adding to the ore charge.

From the point of view of ultimate recoveries, the series of operations employed at Freiberg approaches perfection, but the ore-smelting is by no means up-to-date either as regards labour cost, or, particularly, as regards the comparatively enormous production of foul slag. In this respect, economically as well as metallurgically, there is room for improvement by the substitution of long oblong furnaces of the American type, with syphon-tap lead wells, for the large round Pilz furnaces. The increased capacity thus obtained would involve some saving both of fuel and of labour, whilst the narrowed zone of fusion at the tuyere level would give a higher temperature there, a cleaner separation of matte from slag, and, therefore, a less quantity of foul slag to be re-smelted, as well as a very much lower volatilisation loss of lead and silver.

Analyses of work-lead, matte, speiss, and slag produced at Freiberg are given in Chapter x. (Tables XXIV., XXVI., XXVII., and XXVIII. respectively), while an analysis of flue-dust will be found in Chapter xiii.

**Mechernich.**—The Mechernich ore consists of small spots of galena (and other lead ores) up to pea size, in certain beds of the Triassic (Bunter) sand stone, which are crushed and dressed to get out their metallic contents. The whole of the ore which reaches the smelting works, therefore, consists of "concentrates" of different degrees of fineness. These are roasted in the double hearth reverberatories referred to in Chapter v., and the composition of the slag-roasted ore is given in Table XV. (p. 140).

The roasted ore is smelted in small rectangular blast furnaces together with iron fluxes (puddle cinder and hematite ore) and limestone. Besides a work-lead of great purity containing only 6 ozs. silver per ton (which is afterwards recovered by the Parkes process) a small quantity of matte is formed which is roasted in stalls and added to the ore charge. The composition of the slag, matte, and work-lead is given in Tables XXVIII., XXVI. and XXIV. respectively.

**Goslar (Lower Harz).**—At the Sophienhütte and Juliushütte, ores from the celebrated Rammelsberg mine are treated. The heap roasting of these ores has been described in Chapter v., and the leaching out of zinc sulphate is referred to in Chapter xx. The composition of raw and roasted ores is given in Tables X. and XV.

The ores are deficient in silica for fluxing the bases, and being so high in zinc, even after leaching of the fine portion, require the addition of more iron in order to form the double subsilicate,  $\text{ZnO} \cdot 2\text{FeO} \cdot \text{SiO}_2$ . Both these conditions are satisfied by the use of basic ferruginous slags from the Oker copper works near by, containing 17 to 21 per cent.  $\text{SiO}_2$ , 66 to 70 per cent.  $\text{FeO}$ , and 1 to 2 per cent. Cu. To every 60 tons of roasted and leached fine ore is added 40 tons of unleached lump ore, 25 tons of ferruginous copper slag, and 20 or 30 tons of foul slag from the same process. The water tuyere furnace employed is that figured and described in Chapter viii.; it is blown by five 2-inch tuyeres, supplied with air under a pressure of 10 to 13 ozs., and puts through in twenty-four hours about 10 tons of ore (14 or 15 tons of charge), with a coke consumption of 15 per cent. of the weight of charge.

The products are work-lead with 32 ozs. silver per ton, matte containing all the copper and a small part of the lead and zinc, clean slag which is thrown away, and foul slag (2 per cent. of the whole) containing 5 to 7 per cent. lead

as shots of matte. The composition of the clean slag is given in Table XXVIII. (p. 284). A flue-dust rich in zinc is also produced, the composition of which is given in Table XXXVIII. (p. 336); this is worked up by leaching with sulphuric acid and crystallising out the white vitriol. The matte is once roasted in heaps and re-smelted together with the foul slags from the ore smelting, and fresh copper slag from Oker, yielding a small quantity of enriched matte, work-lead, and a clean slag with only  $\frac{1}{2}$  to  $\frac{3}{4}$  per cent. lead, which is thrown away.

The slags produced at these works are of a most unusual composition. Schnabel quotes\* the following analysis of an exceptional slag produced in 1897, viz. :— $\text{SiO}_2$  12.87,  $\text{Al}_2\text{O}_3$  3.12,  $\text{BaO}$  23.40,  $\text{CaO}$  2.40,  $\text{MgO}$  0.72,  $\text{FeO}$  20.64,  $\text{Zn}$  20.25,  $\text{Pb}$  0.77,  $\text{Cu}$  0.87,  $\text{Mn}$  1.48,  $\text{S}$  6.40 = 92.62. It is not easy to see the mode of combination of the above slag, but probably the Mn was as  $\text{MnO}$ , and the Zn partly as  $\text{ZnO}$ , partly  $\text{ZnS}$ , while the Ba was partly as  $\text{BaS}$ , but chiefly as  $\text{BaSO}_4$ . Recalculation of the available data indicates that the zinc probably existed in the form of 21.31 per cent.  $\text{ZnO}$  and 4.70 per cent.  $\text{ZnS}$ , while the Ba was probably as 18.20 per cent.  $\text{BaSO}_4$  and 12.60 per cent.  $\text{BaS}$ .

**Clausthal (Upper Harz).**†—At Clausthal and Lautenthal is to be seen at work the only surviving example of the old precipitation process, by which raw lead ores in a fine condition are charged into a blast furnace together with iron ores and slags. By the use of a high furnace and a small quantity of low-pressure blast, the temperature before the tuyeres is kept high, while the rate of smelting remains low; sufficient metallic iron is reduced to partially decompose the sulphide of lead, producing metallic lead and iron matte, which takes up the undecomposed lead sulphide. It has been already noted that this reaction generally comes into play more or less in the ordinary blast-furnace reduction process when sulphide lead ores are present in the charge, but at Clausthal it is the sole reaction relied upon. By the slow rate of smelting the consumption of fuel is very high, and the labour cost per ton smelted is also much greater than in the roast and reduction process; while, allowing for the subsequent roasting of the matte, it can hardly be said that the saving of roasting on the ore is a sufficient offset to these disadvantages.

Both circular and oblong furnaces are employed in the precipitation process in the Upper Harz, the latter being a modification of the Raschette furnace, and the former from the Pilz, with a good many modifications. The circular furnace used has been already figured in Chapter viii.; the pressure of blast under which it works is from 7 to 8 ozs.

The average composition of rich dressed Clausthal ore is given in Table X., somewhat poorer ores containing only 50 per cent. lead, and up to 30 per cent. of insoluble residue. The average ore mixture contains, for every 100 tons of this ore, about 200 tons of other material, made up somewhat as follows:—50 tons roasted ore-matte, 50 tons matte-slag (from the next operation), 80 tons ore-slag from the same furnace, 5 tons of leady residues and drosses, and 15 tons of burnt pyrites, tap cinder, and similar iron fluxing materials. Formerly, the quantity of ferruginous slags used was much larger,

\* *Handbuch der Metallhüttenkunde*, vol. i., p. 389.

† Schnabel, *Handbuch der Metallhüttenkunde*.

but now reliance is chiefly placed on roasted matte as a means of supplying the iron required for combining with the sulphur of the ore.

The total quantity of the above mixture passed through a round furnace, of the dimensions shown in Fig. 79, with a blast pressure of 6 to 8 ozs., is about  $10\frac{1}{2}$  to 11 tons, corresponding to only  $3\frac{1}{2}$  tons of ore. The consumption of coke is about 13 per cent. on the whole charge or 40 per cent. on the ore, and three men are required on each twelve-hour shift—viz., one smelter, one feeder, and one slag runner. The slag flows continuously, lead and matte being tapped together from time to time, the matte cake being lifted from the still melted lead as soon as set. For every 60 tons of lead set free about 65 tons of matte are produced. The composition of matte and slag are given in Table XXX.

The working up of the matte consists of a series of alternate roastings and smeltings. The roasting in heaps at Clausthal and in kilns at Lautenthal has been already referred to in Chapter x. The smelting takes place in small square masonry furnaces of antique pattern, with one or two tuyeres in the back. The roasted matte is smelted together with 140 per cent. of ore slag, and the coke consumption is 20 per cent. of the weight of matte or 8 to 9 per cent. of the weight of charge. The small quantity of work-lead produced is richer in silver than that produced direct from the ore, containing 70 ozs. silver per ton as against about 48 ozs. A second roasting and smelting produces a third lead and enriched matte; while a third roasting and re-smelting gives no more lead, but only an impure black copper with 30 per cent. of lead. The two (sometimes three) following roastings and smeltings give further quantities of purer black copper, but the matte continues about the same in percentage—viz., between 57 and 60 per cent. Cu.

The following table shows the weights of lead, matte, and black copper obtained at each successive concentration smelting, beginning with 1,000 tons of ore (to which are added in the ore furnace 500 tons of roasted matte and 1,500 tons of fluxes and slags):—

	Tons Lead.	Tons Matte.	Tons Black Copper.
Ore smelting produces, . . .	600	650	..
1st matte smelting produces, . .	52	150	..
2nd " " " " . . .	9	40	..
3rd " " " " . . .	..	18·4	4
4th " " " " . . .	..	7·3	4·6
5th " " " " . . .	..	4·4	1·4
6th " " " " . . .	often	omitted.	..
Final products, . . .	661	4·4	10·0

The following tables, compiled from data quoted by Schnabel,\* give the analyses of slag and matte obtained from each concentration smelting, the original matte and ore slag being added for comparison :—

\* *Op. cit.*, pp. 406 to 413.

TABLE XXX.—MATTE AND SLAG ANALYSES (CLAUSTHAL).

## MATTE.

	1st or Ore Matte.	2nd Matte.	3rd Matte.	4th Matte.	5th Matte.	6th Matte.
Pb, . . . .	12.44	8.01	13.00	9.06	6.32	3.88
Cu, . . . .	3.49	12.29	35.72	42.30	58.10	59.86
Fe, . . . .	48.69	45.62	21.88	19.98	11.01	12.24
Zn, . . . .	4.31	3.36	2.23	1.59	0.96	0.75
Ag, . . . .	0.035	0.035	0.08	0.075	0.06	0.035
S, . . . .	25.65	21.80	19.49	17.89	20.15	20.78
	94.615	91.115	92.40	90.895	96.60	97.545

## SLAG.

	1st or Ore Slag.	1st Matte Slag.	2nd Matte Slag.	3rd Matte Slag.	4th Matte Slag.	5th Matte Slag.
SiO <sub>2</sub> , . . . .	31.09	24.91	27.21	17.43	17.06	18.97
Al <sub>2</sub> O <sub>3</sub> , . . . .	6.98	4.50	5.85	5.87	5.36	6.47
FeO, . . . .	40.64	51.85	54.56	62.14	65.08	61.84
MnO, . . . .	0.86	0.75	1.07	0.49	0.56	0.50
NiO + CoO, . . . .	0.03	0.13	0.06	0.27	0.34	0.54
ZnO, . . . .	8.01	6.56	6.24	4.30	2.55	2.90
BaO, . . . .	1.04	..	..	..	..	..
CaO, . . . .	4.39	4.25	5.46	2.00	2.32	2.27
MgO, . . . .	0.05	0.38	0.52	0.39	0.24	0.47
K <sub>2</sub> O, . . . .	0.58	0.51	0.64	0.57	0.63	0.77
Na <sub>2</sub> O, . . . .	0.35	0.33	0.31	0.33	0.41	0.53
Pb, . . . .	1.70	1.55	2.25	1.47	1.02	0.92
Cu, . . . .	0.21	0.90	1.39	1.90	1.88	1.58
Ag, . . . .	0.002	0.0022	0.0012	0.0015	undet.	undet.
Sb, . . . .	0.22	0.20	0.08	0.09	0.34	0.20
BaSO <sub>4</sub> , . . . .	0.51	0.30	1.11	tr.	none.	none.
P <sub>2</sub> O <sub>5</sub> , . . . .	0.79	0.69	0.85	0.43	0.45	0.47
S, . . . .	3.55	3.86	2.34	1.47	1.02	0.92
	101.002	101.4922	99.9512	99.1515	99.26	99.35

The above complicated procedure is quoted chiefly on account of its historical interest and as a contrast with modern practice, since none would advocate the adoption of such a series of operations in place of the ordinary modern practice, whereby in only two operations lead matte is converted into a rich copper-silver matte and an impure silver-lead; even allowing for the fact that only a small part of the original matte treated has to undergo the concentrations from No. 2 onward, it is evident that there is unnecessary complication. While at Clausthal there may be possibly something to be said for the retention of the precipitation process in preference to roasting the ore direct, there can be no valid reason why the treatment of matte (up-

to the point where silver extraction commences) should not be cut down to two concentrations instead of five, as above. The second concentration could be made to yield "pimple regulus," as at Freiberg, or a "black copper," if it were preferred to adhere to the Oker process of silver extraction, described in the *Metallurgy of Silver*.

## AUSTRIA-HUNGARY.

Przibram (Bohemia).<sup>\*</sup> This is the largest individual metallurgical centre of the Austro-Hungarian empire. An analysis of the ore treated is given in Table X. The roasting furnaces, twelve in number, are figured and described in Chapter v. An average sample of the ore as roasted contains 34 per cent. lead and 0.260 per cent. (85 ozs. per ton) silver, while, after roasting, the sulphur contents do not exceed 1 per cent., the proportion of copper being so small that there is no need to leave sulphur in the roasted ore to protect it from being carried away in slag.

The roasted ore is mixed with fluxes (tap-cinder, roasted pyrites, and limestone) in the proportion of 37 tons fluxes to 100 tons of ore. It is smelted in seven round water-block furnaces (described in Chapter viii.) with eight 24 inch tuyeres supplied with air at 13 ozs. pressure. In twenty-four hours 22 tons of ore are put through, or a total of 31 tons of charge, the fuel consumption being 16.4 per cent. coke.

The charge carries an average of 16 per cent. lead and 32 per cent.  $\text{SiO}_2$ ; the slag produced averages 1.38 per cent. lead and 36 per cent. silica.

The work-lead produced contains about 150 ozs. of silver per ton. It is liquated before desilverisation in the steam-pattinsonising plant (v. Chap. xv.) in charges of 1 metric ton each, and yields from 4 to 5 per cent. of dross carrying 4 per cent. Cu and 70 per cent. Pb, which removes 80 to 90 per cent. of the copper present. The matte is very small in quantity, and is returned to the ore furnace after roasting in the stalls described in Chapter v. The clean slags containing under  $1\frac{1}{2}$  per cent. lead and  $\frac{3}{4}$  oz. silver per ton are thrown away, a large part being, however, utilised for the manufacture of slag brick. Poul slags containing more than the above-mentioned proportions of lead and silver are returned to the furnaces. Analyses of work-lead, matte, and slag are given in Tables XXIV., XXVI., and XXVIII. respectively, while analyses of speiss and hearth accretions will be found in Tables XXVII. and XXIX. respectively.

## SPAIN.

A large part of the lead produced in this country is from reverberatory and hearth smelting, but several of the largest establishments, notably those at Mazzarrón, at Puertollano, and at Peñarroya, employ blast furnaces exclusively, being mostly circular and of the modified Spanish or German type.

Peñarroya.<sup>†</sup> At these works there are four large brick-built furnaces, one of which is always in reserve ready to blow in. They are rectangular

<sup>\*</sup> Koehnke, quoted in *Min. Industry*, vol. x., p. 430; also Cap. *Oesterr. Zeitschr. f. B. u. H. B.*, 1902, p. 147; and *Min. Industry*, vol. xi., 1902, p. 442.

<sup>†</sup> *Private Notes*, 1908.

with rounded corners, and have nine water tuyeres arranged around three sides. They measure 1.50 m. (5 feet) by 1.80 m. (6 feet) at tuyeres and 6 m. (19 feet 6 inches) in height to feed floor, and are supplied with blast at a pressure of 14 to 18 ozs. The material smelted is chiefly H.H. pot-roasted ore (already partly fluxed), to which is added 10 to 14 per cent. of additional iron ore, or roasted matte, as well as any dross that may happen to be on hand. Of this charge the furnaces smelt each from 100 to 110 tons per twenty-four hours, producing in the same period from 50 to 60 tons of lead, besides a variable amount of matte, say from 3 to 10 tons. The lead carries from 0.13 to 0.14 per cent. silver (say 42 to 45 ozs.), but only the minutest trace of gold; the matte runs 5 to 10 per cent. lead, 1 to 2 per cent. copper, and about 40 per cent. iron. The lead is tapped at intervals into small dressing pots, the contents of which are ladled into moulds as usual. The slag made runs  $\text{SiO}_2$  28 to 30,  $\text{FeO}$  36 to 38½,  $\text{CaO}$  22 to 28,  $\text{Zn}$  2 to 4, and under 1 per cent. of lead. The campaigns usually last from four to six months before the furnace has to be blown out.

A large spray-jacket furnace after the Pertusola model, 7 feet in diameter and over 20 feet high, with fifteen tuyeres arranged in two rows, staggered, with 6 inches difference in height between the rows, has been tried here under a blast pressure of 24 ozs., but has given very unsatisfactory results, the chief difficulty being that with this excessive diameter the blast does not penetrate, and a column of only partially fused material gradually accumulates in the centre of the furnace. It seems probable that such a great diameter can only be successfully employed where the ores are of exceptional fusibility, and where the proportion of fines is low.

**Mazzarron, near Cartagena.**—At this establishment, which turns out over 20,000 tons of lead annually, three large round furnaces are kept running to treat 80,000 tons of ore. They are of the Pilz type, two of them being 6 feet in diameter at the tuyeres, 9 feet at the throat, 18 feet high from tuyeres to feed floor, and having eight 2-inch tuyeres supplied with air under a pressure of 6 ozs. The remaining furnace is oval, 10 feet by 6 feet at the tuyeres, and 29 feet high, with sixteen tuyeres. Sea water is used for the jackets, and the blast is supplied by four No. 3 Root blowers.

The mixture treated is as follows:—Roasted lead ores, 100 tons; irony old slags, 12 to 15 tons; iron ores, 25 to 35 tons; limestone, 20 to 25 tons; slag returned, 40 to 50 tons—total, 210 tons. Of this mixture the two small furnaces smelt about 110 tons each, equal to 50 to 55 tons of ore per day; the large furnace about 140 tons, equal to 65 to 70 tons of ore daily. The furnaces are charged as usual for large round furnaces, coke being thrown to the centre and ore spread round the sides. They are worked as sump furnaces and tapped four times a day, yielding 6 to 8 tons of lead at each tapping, besides a small quantity of matte. The slag runs continuously, and a large proportion of it, containing under 1½ per cent. lead, is clean enough to throw away. Its analysis is given in Table XXVIII.

The condensing flues are 3½ kilometres in length, and the fume produced assays on an average over 50 per cent. lead. Analyses of it are given in Table XXXVIII. It is treated by bricking with lime, sand, and a little fine galena ore, and adding to the ore charge on the last hearth of the roasting furnace, where it is sintered down with the ore.

## ITALY.

**Pertusola.\***—At these works, the largest in Italy, and the original home of the Huntington-Heberlein process, the new blast furnaces are circular, 7 feet 6 inches in diameter at tuyeres and 23 feet high, provided with no less than 15 tuyeres in two tiers, an arrangement which, it is claimed, secures greater capacity concurrently with higher temperature in the smelting zone, better elimination of sulphur, and diminished fuel consumption. These claims, however, hardly appear to be substantiated, the only advantage securable from the placing of tuyeres alternately in two tiers close together in the vertical plane being an approximation to the ideal for a circular furnace of a belt of blast all round the furnace, which should secure greater uniformity in the distribution of the wind, and, therefore, a more perfect utilisation of the available sectional area.

The average tonnage treated per twenty-four hours is 233 tons of a fusible and favourable ore mixture, the average tonnage of lead produced being 73 tons, or 40 per cent. on the charge, with a fuel consumption of 8·04 per cent., or 12·4 tons burden to one of fuel. No matte is made, but the work-lead is carefully drossed in kettles before casting, and yields daily about 44 cwt. of coppery dross containing 40·5 per cent. Pb, 30 per cent. Cu, and 17 ozs. Ag per ton. The composition of the slag produced is given in Table XXVIII.

**Monteponi (Sardinia).†**—At these works the principal ores smelted are of the following classes:—

(1) "Carbonate" ores of lead with much zinc, screened through a 10 mm. screen, the coarse being smelted direct and the fine, after sintering, either alone or with fine galena.

(2) Dry lead ores, mostly quartz with 10 to 15 per cent. lead.

(3) Galena, which is crushed fine and roasted dead, sand with 88 per cent. silica and some alumina and alkalies being added to decompose lead sulphates and frit to silicates. Roasting is effected in two hand-rabbled reverberatories, 59 feet long, which turn out 12 metric tons of roasted ore per twenty-four hours, consuming 1,800 kgs. of English bituminous coal, or 2,400 of Sardinian lignite.

The charge for the blast furnace (already described in Chap. x., Figs. 84 and 85) consists of 50 per cent. roasted and raw ore, 20 per cent. fluxes, and 30 per cent. slag. The fluxes employed are a limestone from the mine 98 per cent. pure, and a limonite from the calamine deposits with 48 per cent. Fe, less than 4 per cent. Zn, a little lead, and traces of copper and silver. Of this charge each furnace smelts 60 tons (=30 tons ore) per twenty-four hours with a blast pressure of 15 ozs., and a fuel consumption of 12 per cent. The slags produced contain SiO<sub>2</sub> 25, FeO 33, ZnO 15, CaO 16, MgO 3, BaO 2, Al<sub>2</sub>O<sub>3</sub> 2·5, Pb 1 to 1½ per cent., and small quantities of alkalies, sulphur, &c. There being but little sulphur in the charge, no matte is produced, but only a large quantity of furnace dross which contains Pb 90, S 1·6, Cu 0·85, Zn 0·4, Fe 0·99, and Sb 0·22 per cent. It is liquated and then reduced in a reverberatory furnace yielding lead and a copper matte, which is smelted in a small blast

\* *Berg-u. Hüttenm. Zeitung*, 1899, p. 189.

† Ferraris, *E. and M. J.*, Oct. 28, 1905, p. 781, translated from *Oest. Zeit. f. B. u. H. Wesen*, 1905, p. 455.



furnace, giving hard (antimonial) lead and a second copper matte with Pb 30, Cu 18, Fe 25, and S 18.4 per cent. This second matte is dead-roasted in reverberatories, sintered, and smelted in the small blast furnace to copper bottoms containing Cu 60 per cent., Pb 25 per cent. These are liquated for more lead, and refined up to blister. All the slags from these subsidiary operations are returned to the ore furnace.

#### GREECE.

**Laurium.\***—At the works of the French Laurium Company a variety of natural carbonate ores mixed in such proportions as to be practically self-fluxing is smelted with roasted lead concentrates obtained from mixed sulphide ores. The fines of the richer ores are screened out to be handled with the roasted lead concentrates, and formerly, up to 1905, they were treated by briquetting with flue-dust; the mixture was made up in layers in 60-ton beds or low heaps about 4 feet high, well watered, and then shovelled into the hoppers of soft mud auger-type brick machines, each with two nozzles, 6 inches  $\times$  4 inches, the prismatic bars being cut off by means of wires so as to make bricks of about 22 lbs. each, which, after drying for two days, were stacked in heaps. These briquettes mixed with calcareous and magniferous carbonate ores formed the smelting charge; the total cost of briquetting was only 1s. 7d. per ton.

In 1906 blast-roasting was introduced for the sulphides and concentrates, superseding briquetting for this material. The preliminary concentration takes place in Oxland furnaces (see Chap. v.), and blast roasting of the product of these together with an equal amount of raw concentrates is carried out in  $1\frac{1}{2}$ -ton pots as described in Chap. vi.

Until 1907 the furnaces were of the old Castilian type, more or less as shown in Figs. 74 and 75; they were twelve in number, the height of the most modern being 14 feet from tuyeres to feed floor, with take off underneath, and the blast pressure averaged only 10 inches of water; the charge averaged a yield of 10 per cent. of lead, and was smelted with 16 to 17 per cent. of poor coke. Each furnace treated about 25 tons of charge per day with a fume loss of 2 to  $2\frac{1}{2}$  per cent. of the charge, or from 20 to 25 per cent. of the total lead present. Owing to the slow rate of smelting, the high reduction, and the low temperature of the smelting zone, great trouble was experienced from accretions both in the hearths and in the shafts of these old-fashioned furnaces.

In 1907 the installation was completed of a large rectangular furnace of the latest type, measuring 48 by 160 inches at tuyeres, which has been already described, together with many of its accessories, in the chapter on *Plant* (see Figs. 95 to 100). The crucible with its special chromite magnesite lining (Fig. 101) and the wrought-iron water jackets, 7 feet high (Figs. 107 to 110), have been already referred to, also the autogenous soldering of the tuyere-sockets into the jackets (Figs. 116 and 117). The stack is built as usual upon arches, the abutments of which are heavy corner brackets with tie rods to take up the stresses. The arrangements for handling matte and slag have been fully described in the same Chapter viii.—namely, the tapping jacket (Figs. 113 to 115), slag spout (Figs. 163 to 167), water-cooled plug for con-

\* *Private notes, 1904, and communications, 1905 to 1908.*

trolling slag flow (Figs. 167 to 169), and the forehearth and slag granulating troughs and hoppers (Figs. 173 to 176).

The work-lead overflows from the lead well into one of two steel cooling pots holding 2,500 kilos., in which it is drossed and tapped at intervals into the usual half-circle of moulds. The matte and speiss are tapped from the torchhearth usually once in twenty-four hours, and instead of the pits formerly employed, the stream is received on long trucks with a row of moulds on them, somewhat after the fashion of the arrangement shown in Figs. 180 to 181, except that the moulds are made in cast iron with much less "flare," and that the trucks are much longer. The granulated slag in particles less than 2 mm. in diameter is received in two 8-ton sheet-iron hoppers, shown in Fig. 176, which are used alternately, one filling while the other is draining and emptying; draining by means of a perforated section at the point of the pyramid takes only five to ten minutes, after which the valve is opened by means of the hand-wheel and the slag is discharged into a waggon below, which holds 8 tons. Two men per shift suffice to handle all the slag from the furnace, besides breaking up and loading the matte and speiss produced, and this when the furnace is smelting from 275 to 300 metric tons of ore per twenty-four hours. The matte-fall is quite low, partly owing to the comparatively thorough roasting of the sulphide concentrates, partly to the rapid smelting and low reduction, by which much residual sulphur is expelled; owing to the high percentage of arsenic in the ores, a good deal of speiss is made, nearly as much as the matte in quantity, but it is uniformly poor, averaging only about 1 per cent. lead and 3 ozs. of silver per ton. Analyses of the various furnace products are given in the various tables in Chapters x. and xiii., but for convenient reference they are repeated here all together.

TABLE XXXI.—ANALYSES OF LAURIUM FURNACE PRODUCTS.

	Work lead.	Slag.	Matte.	Speiss.	Flue-dust.
SiO <sub>2</sub> . . . . .	..	25.20	0.80	0.80	0.16
Fe . . . . .	..	23.24	50.83	63.80	..
Mn . . . . .	..	2.88	1.66	..	tr.
Al <sub>2</sub> O <sub>3</sub> . . . . .	..	11.38	3.83	4.96	..
CaO . . . . .	..	17.33	..	..	..
As . . . . .	0.40	..	0.40	20.75	31.72
Cu . . . . .	0.38	..	1.85	0.63	..
Zn . . . . .	..	9.87	7.57	1.75	3.70
Bi . . . . .	0.09	..	..	..	..
Sb . . . . .	1.60	..	..	..	..
S . . . . .	..	0.78	21.81	4.61	6.43
Pb . . . . .	97.38	0.80	7.75	0.50	39.45
Ag . . . . .	0.150	0.0010	0.025	0.0109	0.003
Au . . . . .	0.0004	..	..	..	..

Blast is supplied at 35 to 40 ozs. pressure by a No. 7 Connersville blower driven direct by a 110 H.P. gas engine.

With a fuel consumption of 13 per cent. of coke, the furnace puts through from 275 to 300 metric tons per twenty-four hours of a self-fluxing charge carrying about 11 per cent. lead, of which 90 per cent. is recovered. The

somewhat aluminous and zincy slags contain from 0·7 to 0·9 per cent. lead. The whole of the work around the furnace takes only 36 men per twenty-four hours, or 12 per eight-hour shift—namely, four on the lower floor and eight on the feed floor; the labour cost amounting to only 1s. or 13d. per metric ton of material handled. The substitution of this modern furnace for the old type formerly in use has resulted, as was anticipated, in the effecting of great economies in every direction, particularly as regards loss of lead, labour cost, and fuel consumption. Plans are now being got out for a still larger furnace, embodying some slight improvements in accessories. The exceptionally large tonnage smelted in these furnaces, which are not abnormal in size, is, of course, a result of the self-fluxing nature of the charge, no portion of which is notably more siliceous than the average.

#### AUSTRALIA.

At the **Broken Hill Block 14 Smelting Works** \* (Port Adelaide, S.A.) four furnaces were formerly in blast, treating an ore charge of unique composition. About three-fourths of the ore treated, or two-fifths of the total charge, including fluxes, was composed of roasted concentrates and slimes from the concentrating plant of the Company's mines at Broken Hill. The former contained 55 to 60 per cent. Pb, 25 to 30 ozs. Ag, and 8 to 10 per cent. Zn; the latter, 30 to 35 per cent. Pb, 20 to 25 ozs. Ag, and 15 to 20 per cent. Zn.

These concentrates and slimes were mixed in any convenient proportions according to the stocks on hand, and roasted in the three-hearth reverberatories described in Chapter vi. No attempt was made to roast below 7 to 10 per cent. S, as it was found that, by using a large volume of blast so as to keep the furnace gases oxidising, or at least not distinctly reducing, the reactions so well known in connection with reverberatory smelting come into play, and reduce the lead just as well as if the roast had been complete and reduction were effected as usual by means of CO and glowing carbon. The ore was, therefore, rushed through the roasters at the rate of nine charges in twenty-four hours (an unusual procedure for such material), and the temperature on the last hearth was just high enough to bring about incipient softening and to form lumps. There were thirteen of these roasters in operation, all of uniform type; there was no appreciable volatilisation loss; and the flue-dust formed was trifling in amount. Further details about the roasting process are given in Table XI.

The remainder of the ore treated consisted of siliceous and ferruginous carbonates, with about 10 per cent. lead and 10 ozs. silver. The fluxes used were barren iron ore with 15 per cent. silica, costing 10s. 6d. per ton; and limestone (96 per cent.  $\text{CaCO}_3$ ), costing 6s. 3d. per ton; the fuel being English coke (7 per cent. ash), costing 29s. to 33s. per ton, all delivered at the works. The furnaces employed have been described and figured in Chapter viii., Figs. 91 and 92, but their manipulation was attended with some special difficulties on account of the high zinc contents. The "hangings" gave great trouble until it was discovered that "drawing in" the furnace from immediately above the jackets to the bottom of the downtake by only 6 inches

\* *Private Notes*, 1897.

or so, all round the furnace, thus giving a slight overhang to the brickwork, was of very great efficacy in preventing their growth.\* The large amount of fume given off caused trouble to the feeders when the ordinary open tops were used. This could have been got over by using a stronger chimney draught, at the expense, however, of less condensation and higher losses. More recently, a large ventilating hood in the roof was provided at a considerable height over each furnace, and connection was made with this by means of a sheet-iron chamber provided with large feed-doors at each side, and running on wheels so as to be easily run back for barring the furnace.

Each furnace was run by its own independent blower, a plan which, as already seen, possesses many advantages, especially when the ores are not particularly docile. The tuyeres of these furnaces being almost absolutely air-tight, and the loss by "slip" on ordinary types of impeller blower, although large when running against a high pressure, comparatively small when running against a low pressure, it was easy to calculate the ratio between the quantity of air delivered into the furnace and the fuel consumed. Allowing for slip and leakage, the amount of air delivered into the furnace was in excess of that required to burn all the coke to  $\text{CO}_2$ ; the atmosphere inside the furnace, therefore, may have been oxidising, but can certainly not have been reducing in its effects.

Each furnace was provided with two syphon taps on the same side, terminating in small lead wells. These were tapped alternately at short intervals into small cooling pots, where the lead was allowed to cool, and was carefully "drossed" before ladling into the usual series of reversible moulds hung upon frames, the abundant dross (chiefly PbS) being cast into two or three bars which were returned to the furnace.

The column of charge was run quite exceptionally low, usually about 8 feet below the feed floor, making the stock line only about 7 feet above the centre of the tuyeres. The down-takes of all four furnaces led to a single flue 7 feet wide, 12 feet high, and 400 feet long, in which 6 per cent. of the total weight of ore smelted was collected. The stack at the end was 11 feet in diameter and 120 feet high. Slag taps † and spouts were provided at each end of the furnaces and were used alternately, so as to keep both ends as far as possible under equal conditions. The furnaces were, of course, placed lengthwise of the building, instead of transversely, in order to admit of this.

Part of the slag made, after running through an overflow pot, was taken out in small hand pots and utilised for extending the works, which are built upon a reclaimed foreshore, but the bulk of it was handled in large single-pot trucks provided with tapholes for running out the molten interior, the whole of the slag shells from both classes of pot (amounting in the aggregate to something like 20 per cent. of the total slag made) being returned to the furnace. The composition of the slag is given in Table XXVIII.; it contains a large amount of  $\text{ZnS}$ , which burns and smokes so much at the taphole and in the pots that a separate ventilating chimney, 5 feet in diameter, has to be run up over each slag tap to take off the fumes. With the exception of this inconvenience, and of the necessity for keeping a specially close watch upon the

\* This reduction in the width of the furnace immediately above the jackets is not shown in the Figs., having been introduced since they were drawn; it has proved to be of the greatest value.

† i.e., matte-separating taps like those shown in Figs. 146 to 153.

running of the furnace, regular descent of charges, breaking up of incipient "scaffolds," and good condition of the tuyeres, very little difficulty is caused by the high zinc contents of the charge. The slag produced is hot and runs very thin; unlike the more siliceous slags, however, which pass through a plastic stage, it sets suddenly on cooling, and becomes quite hard and stony while still at a bright red heat.

For anything like good work the composition of these slags, containing 14 to 15 per cent. of metallic zinc, must be kept within very narrow limits as regards both silica and lime, the limits for the former lying between  $23\frac{1}{2}$  and 26 per cent., and for the latter 12 to 16 per cent. If the slag contains less than 10 per cent. lime, it carries away matte in suspension and the losses are increased; if there is more than 16 per cent. lime, the slag becomes mushy and gives rise to bad, irregular smelting. If the silica falls to 21 or 22 per cent., the losses of lead and silver in the slag are at once largely increased; if, on the other hand, it rises above 26 per cent., the lead loss increases rapidly without a corresponding increase in the silver loss, which, in fact, is generally lower on account of the larger amount of matte made. The non-formation of matte when smelting an ore charge containing 6 to 8 per cent. of sulphur and producing a normal slag, and its immediate appearance as soon as the silica contents of the slag rise above the normal limit without any change in other conditions, is one of the most remarkable facts brought out by the novel work of T. J. Greenway on these zinc ores. There seems to be no doubt whatever about the fact, which has been repeatedly observed whenever the ratio of silica to iron in the charge increased either by accident or design. The explanation is by no means obvious, but a clue is afforded by the fact that the matte so formed is not the ordinary iron matte, but one very high in lead and zinc, and containing only a small percentage of iron. Further, it has been noticed that a rise in silica is always accompanied by a rise in lead, even though the silver contents of the slag remain low. The practice of running slags high in ZnS is dead against experience elsewhere, and appears to be only justifiable under the peculiar conditions for which it was designed—namely, a bullion fall of something like 20 per cent. with low silver contents—so that the loss of that metal in slags was not above 1 to  $1\frac{1}{4}$  ozs. per ton, even with 2 to  $2\frac{1}{2}$  per cent. lead, high cost of fuel and fluxes, impossibility at the time of diluting the zincy ores by others free from that metal, and absence of copper with which to make a good matte for slag-cleaning. Under ordinary custom smelting conditions zinc in slags should never be run above 12 or 13 per cent. at the outside (as much lower, of course, as possible), especially when alumina is also high, and the presence of ZnS in particular should be avoided by carefully roasting all ores high in zinc.

At the **Broken Hill Works** \* of the **Broken Hill Proprietary Company**, siliceous and ferruginous carbonate ores (containing some galena) were formerly smelted on a large scale in fifteen of the so-called 80-ton furnaces similar to that shown in Figs. 91 and 92, but on a smaller scale, being only 112 inches by 60 inches. The quantity of ore treated per day was about 48 tons per furnace, exclusive of lime and returns. The average yield of the 120,484 tons smelted during 1896 was 7.76 per cent. Pb and 21.5 ozs. Ag, the 9,357 tons work-lead produced thus averaging 207 ozs. per ton. Besides the lead, 1,406 tons of matte were produced (or 1.16 per cent. on the ore)

\* *Private Notes*, 1897.

carrying 13 per cent. Cu and 114 ozs. Ag per ton, all of which were shipped to the refinery at Port Pirie to be worked up. The limestone flux used (92 per cent.  $\text{CaCO}_3$ ) cost 14s. per ton, and the coke (9 per cent. ash) 57s. 6d. to 60s. per ton, delivered at the works. Owing to the high cost of fuel and flux, a siliceous slag was run which was supposed to increase the furnace capacity, at the expense, however, of much higher furnace losses.

The slag produced was run off from the furnace continuously through the matte-separator, or internal forehearth already described (Chap. viii., p. 213), into the large double-pot trucks, also referred to, which were drawn by horses to the dump. The lead produced being small in amount was only tapped from the lead well about once, or at most twice, in a shift. A hemispherical cast-iron ladle with spout attached was so arranged as to revolve upon a pillar in front of the furnace; the end of the spout thus traversed in succession a row of reversible moulds placed upon a frame in front of the furnace, so as to cast the lead direct from the well without any ladling or drossing. As the refinery belongs to the same company, and the dross has ultimately to be worked up into copper matte, no inconvenience resulted from this economical practice.

The furnaces were fed through "open tops," and the column of charge in the furnace was kept 5 feet below the feed floor, leaving the "downtake" free; each plant of furnaces (nine in the north plant, six in the south) was provided with a separate flue and a stack, 180 feet high by 10 feet diameter, of sheet iron lined with one thickness of brick. The flue space provided was, however, utterly inadequate to condense the enormous amount of fume produced, and the volumes of dark grey smoke issuing from the stacks proved the losses of lead to be unusually heavy for an important smelting plant. In criticising this heavy loss it should be borne in mind that the *net* value of lead in flue-dust at Broken Hill, allowing for treatment and realisation charges, is much smaller than at American or European smelting works, and, in fact, until quite recently, lead was about the cheapest thing that Broken Hill metallurgists had to handle.

More recently the whole of the smelting operations of the B. H. P. Co. have been transferred from Broken Hill to Port Pirie, the reasons being the high cost of water at the Hill and the high railway rates on fuel and fluxes. So long as rich carbonate ores were available they could be smelted at the mines at about the same cost as on the coast, but when it came to roasting sulphide concentrates the cost of fuel for that operation became prohibitive, and the smelting plant at Broken Hill was accordingly abandoned.

At Port Pirie\* roasted concentrates and heap-roasted slimes (*v.* Chap. v.) from the dressing works at Broken Hill are smelted with a certain small amount of raw concentrates in eight large furnaces, 62 by 212 inches at tuyeres (described and figured in Chap. viii., Figs. 93 and 94), with central hood for removing fumes. The charge consists of pot-roasted concentrates 2,000 lbs., sintered slimes 1,000 lbs., raw concentrates 200 lbs., old slag 800 lbs., iron ore 1,050 lbs., limestone 550 lbs.; total, 5,600 lbs., which is smelted with 840 lbs. coke or 15 per cent. on the charge. It should be borne in mind that the pot-roasted concentrates which make up the bulk of the charge are already partially fluxed (see Chap. vi.), hence less flux is required to be charged as a separate item of the smelting charge than was formerly the case. The

\* Delprat, *Trans. Aust. I.M.E.*, vol. xii., 1907, pp. 1-29.

charge averages about 17 per cent. Pb, of which 95 per cent. is recovered, together with 98 per cent. of the silver; each furnace produces about 25 to 30 tons of lead per twenty-four hours. Blast is furnished by eight Green blowers of 9,400 cubic feet capacity, each at 30 ozs. pressure, run by three engines aggregating 1,150 H.P. The composition of the slag and other details will be found in Table XXXIII.

At Fremantle (W. A.)\* a three-furnace plant is at work, principally on rich gold ores from Kalgurli, with some local lead and other ores, the chief leady material available, however, being Broken Hill concentrates.

The furnaces are  $52 \times 112$  inches at tuyeres, with cast-iron jackets 4 feet high, having openings between each pair of jackets for water tuyeres projecting 8 inches into the furnace. The downtake is under the feed floor, the height to which is 17 feet, giving an effective height of 14 feet for the charge column; the gases are drawn off through a main flue 350 feet long to a main chimney, 150 feet high. The lead concentrates being only plain roasted, and part of the gold ore fine, there is a good deal of flue-dust made, assaying up to 30 per cent. lead, 2 ozs. silver, and from 0.2 to 0.325 oz. gold. The bulk of the ore treated is rich Kalgurli ore containing on an average 50 per cent. silica, 14 per cent. alumina, 5 per cent. pyrites, with 7 per cent. FeO and 10 per cent. CaO as carbonates, together with from 3 to 5 ozs. of gold per ton. When lead ores were plentiful and a bullion fall of 12 to 15 per cent. on the charge could be secured, the average slags made contained 1.75 per cent. lead (by wet assay), 0.43 oz. silver, and 0.027 oz. gold per ton; but when lead was scarce and the bullion fall was only 7 per cent. with a matte fall of  $4\frac{1}{2}$  per cent. (the normal condition of affairs), the slags made carried  $2\frac{1}{2}$  per cent. lead (by wet assay), 0.8 oz. silver, and 0.05 oz. gold. The average make-up of the charge is as shown in Table XXXIII, which also gives a great many data in regard to the practice carried on.

English coke cost d/d at the works from 52s. 6d. to 57s. 6d. per ton, but a good local non-bituminous and non-caking coal being available from Collie (W. A.) at one-third of the price, it was used to replace 20 per cent. of the normal coke charge, by which means not only was some saving effected but the furnace ran faster, and there was less tendency to the formation of wall accretions.

The bullion made averaged 20 to 30 ozs. of gold and 60 to 70 ozs. of silver, and the direct recovery of gold in bullion averaged 95 per cent. of the total gold charged, the extremes registered with very low and high proportions of lead on the charge being 87 and 99 per cent. respectively. Owing to the fact that payment was not made for the silver contents of the gold ores, the apparent recovery of silver was from 98 to 100 per cent. The matte made contained on an average 16.5 per cent. of lead, 7 ozs. of silver, and 0.28 oz. of gold per ton. It was crushed as usual and roasted in hand-rabbled reverberatory furnaces at the rate of 12 tons per day per furnace, and at an inclusive cost of 11s. 6d. per ton.

Iron flux of good quality cost 15s. per ton delivered at the works, limestone of poor quality cost 4s. per ton. The total cost of smelting (details of which are given in Table XXXIV.) amounted to £1, 12s. 6d. per ton of ore smelted, the unusually large proportion of fluxes required (94 per cent. on

\* *Private communication from L. Pitblado, 1904.*

the weight of ore) accounting for the exceptionally high cost. Further details as to the furnace works are given in Table XXXIII.

The high proportions of alumina and zinc in the charge, combined with the practical absence of copper and the low bullion fall owing to scarcity of lead, render the circumstances in general about as unfavourable as possible for the recovery of precious metals. The Fremantle practice described is altogether abnormal, and is quoted as an instance in which smelting methods have triumphed over unsuitable conditions rather than as an example of custom smelting practice of a normal character.

At Dapto (N. S. W.)\* very similar work was done a few years ago when the output of rich gold ores from Kalgurli was large, and details are given in column 4 of Table XXXIII. The figures there given refer to a period when the gold in the furnace bullion was exceptionally high. On some occasions the lead bullion was run up to even 60 and 80 ozs. of gold per ton, and even then the slag losses were remarkably small (see Table XXXVI., Chap. xii., for details of slag losses when producing exceptionally rich bullion). It may be noted that such rich furnace bullion was only produced when treating a considerable proportion of rich Kalgurli screened fines, often containing over 100 ozs. of gold per ton. Such fines, of course, arrived at the works in bags, and it was found best to charge them into the furnace in the original bags, adding a bag at a time at regular but considerable intervals. The slags made varied in gold assay from 0.005 oz. with 10 ozs. bullion up to 0.06 oz. with 60 ozs. bullion, but all slags made with more than 0.025 oz. gold were saved and re-smelted. The author understands that the Dapto plant has been rebuilt at another point, and recent practice at the rebuilt plant is more nearly upon the general lines of smelting work in the U.S. and Mexico.

#### CANADA.

At the Sullivan Smelting Works (E. Kootenay, B.C.) mixed sulphide ores with Pb 27 per cent., Zn 12 per cent., Fe 18 per cent.,  $\text{SiO}_2$  7 per cent., and CaO 3 per cent., with 11 ozs. As per ton, are mixed with 10 per cent. of siliceous ore and blast-roasted (*v. Chap. vi.*). The blast-roasted ore is smelted in two furnaces, each 42 by 128 inches, with seven tuyeres a side, supplied by a Connersville blower of 68 cubic feet per revolution with blast at a pressure of 18 ozs. Each furnace puts through per twenty-four hours 120 tons ore charge and 30 tons slag, or 3.2 tons total burden per square foot of hearth area, with a coke consumption of 11 per cent. No matte is made, but a good deal of dross (chiefly PbS) accompanies the lead. The slag produced contains  $\text{SiO}_2$  20 per cent., FeO 26 per cent., CaO 10 per cent., MgO 4 per cent.,  $\text{Al}_2\text{O}_3$  4 per cent., Zn 15 per cent., with about 1 per cent. Pb and under 1 oz. of silver per ton.

At Trail (B.C.)‡ the lead smelting plant consists of two blast furnaces, 45 by 140 inches and 45 by 160 inches respectively, which treat, the former 125, and the latter 150, tons per twenty-four hours, the lead ore having been previously roasted in four 26-foot Heberlein furnaces, and agglomerated in 24 H.H. pots.

\* *Private communication*, E. A. Weinberg, 1904.

† *Min. Ind.*, vol. xv., p. 532.

‡ *Canadian Mining Journal*, 1907, vol. xxviii., p. 421.



Most of the smelting done in Canada is, however, matte smelting, and is, therefore, not further described here.

## UNITED STATES.

Since the consolidation of smelter industries in the States by the American Smelting and Refining Co., it has been the policy of that company not to permit the publication of information in regard to the plants, and although the author has had the privilege of going over several, it has been on the express condition that no complete specific descriptions were to be published. Apart, therefore, from such data as have been given incidentally in the preceding chapters illustrative of modern practice, which, originating in the United States, has become standard in other countries, and has even spread to Europe, the only data which can be given here are such as have been already published in technical periodicals.\*

Apart from differences in general arrangements caused by the character of the prevailing ore supply or by the lay out of the works, the actual furnace practice at the great American custom plants shows less variation than at any former time, having become more or less standardised by elimination of details which have not stood the test of practice. As regards the very important feature of the composition of slags aimed at, Ingalls gives † the following table showing the average composition of the slag made at some of the leading American smelting plants during the month of October, 1901 :—

TABLE XXXII.—SLAGS PRODUCED AT AMERICAN SMELTING PLANTS.

Name of Plant.	Ag. Oz. per ton.	Pb. Per cent.	SiO <sub>2</sub> . Per cent.	FeO. Per cent.	MnO. Per cent.	Zn. Per cent.	CaO. Per cent.	MgO. Per cent.	BaO. Per cent.	Al <sub>2</sub> O <sub>3</sub> . Per cent.
Globe, Denver, .	0·46	0·38	31·3	26·2	8·0	5·1	14·5	2·5	0·7	6·9
Graut, „ .	0·86	0·43	30·2	27·5	6·7	5·9	14·8	..	..	6·4
Pueblo, Pueblo, .	0·90	0·80	30·9	26·6	9·0	4·0	14·2	1·2	2·4	6·2
Eilers, „ .	0·88	0·88	31·4	25·6	10·7	5·7	14·2	1·2	0·9	6·7
Philadelphia, „ .	0·80	0·80	30·8	31·1	5·3	5·6	12·3	1·3	1·2	6·9
Durango, Durango, Arkansas Valley,	0·83	1·11	30·7	32·5	..	5·8	19·4	..	1·3	..
Leadville, . .	0·80	0·90	31·5	31·6	6·3	5·3	14·8	2·6	1·2	4·5
Germania, Salt Lake, . . .	0·56	0·84	37·6	27·0	2·7	5·1	20·8	..	..	..
East Helena, Mon- tana, . . .	0·70	0·96	34·7	28·1	2·0	6·9	20·4	1·4	..	3·0
Silver City, . .	0·20	0·50	36·4	30·2	0·8	4·3	15·1	1·0	..	8·1

The above figures may be compared with the average slag recommended by Iles—viz., 32 per cent. SiO<sub>2</sub>, 34 per cent. FeO, and 20 per cent. CaO—and with that from the Murray plant described on a subsequent page, and it

\* See Ingalls, *Lead Smelting and Refining*, 1906, pp. 285-305, reprints of articles in the *E. and M. J.*, April 19, 1902; June 28, 1902; and translations of articles by O. Pufahl in *Zeits. f. B. H. u. Salinen Wesen im Preuss. Staate*, 1905, vol. iii., pp. 400-444; also *Min. Ind.*, vol. xiv., pp. 397 and 398; vol. xv., p. 532; vol. xvi., pp. 667 and 668.

† *Lead and Zinc in the U.S.*, New York, 1908, p. 45.

will be seen that there is less difference in the composition of these slags than in those given in Table XVII., Chap. vii., which represent the limits of good practice under more extreme conditions.

Table XXXIII. shows some not very recent data from two representative American works side by side with similar figures from Mexican and Australian works, and specific references in regard to the roasting and smelting practice at other works have been given in Chapters v., vi., viii., ix., and x. The following abstracts of data, which have been already published in regard to other plants, may be given here.

At the Pueblo plant of the A. S. and R. Co. (Pueblo, Colo.)\* there are seven blast furnaces, 48 by 148 inches at tuyeres, with 18 tuyeres supplied with blast at 34 ozs. pressure, each of which smelts per twenty-four hours about 150 tons of charge, producing 11 to 15 tons of bullion, and a somewhat less tonnage of matte carrying 8 to 12 per cent. each of Pb and Cu. The mechanical charging arrangements for feeding the furnaces with 4-ton charges have been described (Chap. viii.), as has also the roasting of ore and matte in Bruckner and O'Hara furnaces (Chap. v.). The roasted matte is concentrated in a separate furnace with siliceous copper and gold ores to a second matte of 45 to 50 per cent. Cu, which is shipped to the refining plant. The flue-dust from the furnaces is briquetted in White presses (v. Chap. ix.). All the lead bullion produced is refined at a refinery attached to the smelting works, which is where the Howard press, stirrer, and skimmer were first introduced (see Chap. xvi.).

At the Eilers plant (Pueblo, Colo.)† there are six furnaces, 48 by 164 inches, which smelt 160 to 180 tons per day each, with a coke consumption of 11 per cent. on the charge, a bullion fall of 10 per cent., and a matte fall of 8.3 per cent. All the sulphide ores received (except rich Idaho lead concentrates which are smelted raw) are roasted down to 5 to 7 per cent. sulphur in 15 hand-rabbed reverberatories, which put through about 15 tons per day each. The furnaces are hand-charged, and the slag is handled by means of forehearth and tapping pots, the shells from which (30 per cent. of the total slag) are returned for re-smelting. The matte is roasted and concentrated in a separate furnace, as at Pueblo, and the concentrated matte with 45 to 50 per cent. copper is sent to the *Philadelphia* works, which has become a copper refining plant.

At the Globe plant (Denver, Colo.) there are seven furnaces, 42 by 144 inches, with an effective height of charge column of 18 feet 6 inches, each provided with 16 tuyeres supplied with blast at 32 ozs. pressure, and each smelting 120 to 150 tons of charge per day. Double Nesmith pots are used for taking matte and slag to the reverberatory settling furnaces already described in Chap. viii. Flue-dust collection is referred to in Chapter xiii; matte concentration in Chapter x.

At Murray‡ (Utah) there are eight blast furnaces, 48 by 164 inches, with a 20-foot charge column and water jackets 6 feet high, with ten 4-inch Eilers tuyeres per side supplied with blast at 34 ozs. pressure. The large amount of sulphide ore received by this plant is mostly roasted in Bruckner cylinders,

\* *Private Notes*, 1902, and *Min. Ind.*, vol. xiv., p. 398 (from Pufahl, *Zeits. f. B. H. u. S. W. im Pr. Staate*, vol. liii., p. 439).

† Pufahl, *loc. cit.*, p. 439; abstract in *E. and M. J.*, May 12, 1906, p. 889.

‡ *Min. Ind.*, xiv., 397; and xv., 667.

TABLE XXXIII.—EXAMPLES OF LEAD BLAST-FURNACE SMELTING.

Reference.	1	2	3	4	5	Pennsylvania Sm. Works, Utah, 1896.	Germania Sm. Works, Utah, 1897.	Monterey, Mexico, 1902.	Agua Calientes, Mexico, 1902.
Number of furnaces in blast,	1	4	8	..	3	4	4	8	3
Dimensions at tuyeres,	48" × 160"	60" × 160"	62" × 212"	56" × 172"	52" × 212"	45" × 140"	45" × 140"	42" × 160"	42" × 128"
Height tuyeres to feed floor,	20' 4"	14' 8"	21' 9"	..	17 ft.	20 ft.	20 ft.	20 ft.	17 ft.
Height of charge column,	20 ft.	11' 8"	21 ft.	20 ft.	14 ft.	20 ft.	20 ft.	20 ft.	..
Downtake overhead, under floor,	..	..	..	..	..	..	..	..	..
&c.,	..	..	..	..	..	..	..	..	..
Number and diameter of tuyeres,	overhead 20 of 3½"	underneath 18 of 3½"	central 22 of 3½"	overhead 16 of 3½"	underneath 20 of 3"	overhead 14 of 3½"	overhead 14 of 3½"	overhead 16 of 3½"	overhead 16 of 3½"
Description of tuyeres,	flush	projecting	flush	project. 10"	project. 8"	flush	flush	flush	flush
Total area of tuyeres, sq. in.,	1924	173.2	211.6	153.9	140.8	134.7	134.7	133.3	..
Hearth area at tuyeres, sq. ft.,	53.3	44.4	91.3	67.1	76.5	43.75	43.75	43.66	..
Hearth efficiency, tons, 2000 lbs.,	5.67 to 6.20	2.1 to 2.3	1.85 to 2.20	2.23	2.18	3.0 to 3.7	3.6 to 4.1	..	..
Tuyere efficiency "	1.67 to 1.72	0.54 to 0.59	0.76 to 0.91	0.97	1.18	0.97 to 1.20	1.20 to 1.33	..	..
Average blast pressure employed, ozs.,	35 to 40	12 to 14	30	20	16	..	..	40	40
Weight of charge :—	..	..	..	..	..	..	..	..	..
Carbonate ores,	..	250	..	500	500	400	..	..	..
Roasted lead ores,	..	850	..	..	..	200	..	..	..
Pot-roasted concentrates,	..	..	2000	..	..	..	1100	..	..
Sintered slimes,	..	..	1000	..	..	..	..	..	..
Raw sulphide ores,	..	..	200	..	..	300	..	..	..
Siliceous ores,	..	..	..	500	2150	..	..	..	..
Miscellaneous ores,	..	..	..	250	..	600	about 300	..	..
Iron flux,	..	400 to 500	1050	150	1120	..	..	..	..
Limestone,	..	200 to 100	550	200	1370	250	about 100	..	..
Foul slag,	..	200 to 250	800	150	460	..	..	..	..
Roasted matte,	..	..	..	250	400	..	..	..	..
Total,	5290	2000	5600	2000	6000	1750	1500	..	..
Weight of coke per charge,	688	220	840	200	600	200	..	..	..
Percentage of coke on charge,	14%	11%	15%	10%	10%	11½	12 to 13	12	12
" " on ore alone,	14%	..	..	13%	20%	..	..	..	..

TABLE XXXIII.—Continued.

Reference,	1	2	Port Adelaide, S.A., 1896.	Port Pirie, S.A., 1906.	4	5	6	7	8	9
Laurium, Greece, 1908.	275 to 300	95 to 110	20 to 30	150 to 175	130	145	130 to 160	160 to 180	135 to 165	110 to 120
Tl. wt. of charge per 24 hrs., tons,	..	..	..	17	12% 12%	7 % 41%	18 to 20	11% 10%	12% 5%	11 to 12%
Average bullion fall, ..	..	..	..	..	..	..	..	..	..	..
" matte fall, ..	..	..	..	..	..	..	..	..	..	..
Composition of ore charge :—	..	..	..	..	..	..	..	..	..	..
Lead, .. %	..	..	..	..	16	8	..	..	..	..
Copper, .. %	..	..	..	..	1	..	..	..	..	..
Silver, .. %	..	..	..	..	10	..	..	..	..	..
Gold, .. %	..	..	..	..	4	..	..	..	..	..
Wt. of bullion produced per 24 hours, .. tons	..	15 to 20	..	25 to 30	..	..	..	15 to 17	15 to 18	..
Composition of bullion :—	..	..	..	..	..	..	..	..	..	..
Silver, .. ozs. per ton	49	..	..	90	16-5	60 to 70	..	..	400	265 to 400
Gold, .. " "	0-13	..	..	30	..	20 to 30	..	..	..	..
Wt. of matte produced per 24 hours, .. tons	..	none	..	..	..	..	5 to 6	14 to 16	..	..
Composition of matte :—	..	..	..	..	..	..	..	..	..	..
Lead, .. %	8	..	..	..	15	16-5	12	12 to 15	..	..
Copper, .. %	2	..	..	..	4	..	10	8 to 16	..	9 to 15
Zinc, .. %	7 to 8	..	..	..	6	..	..	..	..	..
Silver, .. %	8	..	..	..	20	7	40	130 to 200	..	..
Gold, .. " "	..	..	..	..	0-20	0-28	..	..	..	..
Composition of slag, SiO <sub>2</sub> , %	25	23-5 to 26	..	..	33	39	33 to 36	..	36	34 to 35
" FeO and MnO, %	33	32 to 34	..	..	35	26	30 to 33	..	32	32 to 33
" CaO and MgO, %	17 to 18	13 to 15	..	12	17	15	18 to 23	..	18	18 to 20
" Al <sub>2</sub> O <sub>3</sub> , %	11 to 12	2 to 4	..	6	5	12	3 to 4	..	..	..
" ZnO, .. %	12 to 13	15 to 20	..	13	6	4	3 to 5	..	..	..
" S, .. %	0-8	..	..	3	..	..	..	..	..	..
" Pb, .. %	0-80	2-5 to 3-5	..	1-5	1 to 1-2	2-5	0-5 to 0-75	..	1-25	1-53
Percentage of flue-dust saved, ..	..	6	..	3-3	3	3	2 to 3	..	..	..
Composition of flue-dust, Lead, %	..	25 to 30	..	..	30	30	20 to 25	..	..	..
" Silver, ozs. per ton	..	29/ to 33/	..	..	1 to 2 ozs.	55/	20 to 25	..	25/ to 28/	25/ to 28/
Cost of coke per ton of 2240 lbs., Ton, to which all above figures refer to, .. ozs. per ton	2204 lbs.	2240 lbs.	2240 lbs.	2240 lbs.	0-75	0-8	2000 lbs.	2000 lbs.	2204 lbs.	2204 lbs.
Silver, .. " "	0-33	1 to 1 1/4	..	under 1	0-02	0-03	0-6 to 0-8	..	1-25 to 1-5	1 1/4 to 1 1/2
Gold, .. " "	..	..	..	..	..	..	..	..	..	..
Method of separating slag from matte, ..	Large fore-hearth.	Tapping pots, all	..	..	..	Tapping pots, all	..	..	Forehearth and rever-	Two fore-hearths

the working of which is described in Chapter v. The charge is 8,000 lbs., including 30 per cent. of slag shells as at the Eilers plant and, with a coke consumption of 12 per cent., a bullion fall of 13 to 14 per cent., and a matte fall of 10 per cent., the furnaces prior to the introduction of blast-roasting put through an average 168 tons of charge per day apiece. Since the adoption of blast-roasting (v. Chap. vi.), the tonnage put through has been increased to 200 tons per furnace per day. The slag is handled by means of forehearth and large tapping pots, all the shells from which are saved; its average composition is  $\text{SiO}_2$  36 per cent.,  $\text{FeO}$  29.6,  $\text{CaO}$  23,  $\text{Zn}$  3.8,  $\text{Al}_2\text{O}_3$  4 per cent.,  $\text{Pb}$  0.6 to 0.7 per cent.,  $\text{Ag}$  0.7 to 0.9 oz. per ton. The large quantity of first matte produced (with 15 per cent.  $\text{Pb}$  and 10 to 12 per cent.  $\text{Cu}$ ) is roasted in 12 hand-rabbed reverberatories, 60 by 16 feet, each of which puts through 14 tons ore or 13 tons matte per twenty-four hours, roasting down to  $3\frac{1}{2}$  to 4 per cent.  $\text{S}$  with a coal consumption of 32 to 34 per cent. The roasted matte is concentrated as usual to a concentrated matte of 12 per cent.  $\text{Pb}$  and 45 per cent.  $\text{Cu}$ , as mentioned in Chapter x. The collection of flue-dust at this plant is referred to in Chapter xiii.

At the Bingham \* plant of the U.S. S. and R. Co. there are six furnaces, 45 by 160 inches, with ten 4-inch tuyeres per side, the only peculiarity of which is the elliptical wrought-iron shell surrounding the crucible. The charge column is only 13 feet in height, which, because of its open character, is found sufficient, and the furnaces put through an average of 170 tons per twenty-four hours each, equal to 3.4 tons per square foot of hearth area. The slag is handled by means of forehearths, followed by large Nesmith tap-hole pots, all the shells from which are saved and re-smelted.

At the works of the Cherokee Lanyon Sm. Co. (Kansas) † residues from zinc distillation are smelted together with galena, iron ore, and limestone in a furnace 36 by 90 inches at tuyere line, with 10-inch side- and 6-inch end-bosh, and five 3-inch tuyeres on each side. The charge column was at first only 11 feet, and the saving of lead was poor, but by raising it to 14 feet the recovery was increased to over 90 per cent.

The residues, after weathering and screening from coarse coke, contain 10 per cent.  $\text{Pb}$ , 7.5 per cent.  $\text{Zn}$ , 38 per cent.  $\text{Fe}$ , 10 per cent.  $\text{SiO}_2$ , 1 per cent.  $\text{Cu}$ , 10 ozs.  $\text{Ag}$ , and 0.03 oz.  $\text{Au}$ . They are mixed with galena, slag, limestone, and iron ore when necessary, in such proportions that the residue usually forms 35 per cent. and slag about 32 per cent. of the total charge, which is smelted with 13 per cent. of coke. Trouble is caused in the crucible through the high reduction resulting from the coke in the residues, while, when semi-anthracite is used in the distillation, the lead saving in the blast furnaces drops still lower, owing to hot top. The method of feeding is to spread most of the lead ore round the sides, sending the lead residues to the centre together with the lime flux, and the slag evenly all over. The furnace puts through 46 tons per day of a charge, the bullion fall of which is 9.1 per cent., and the matte fall 7.2 per cent. The slag produced runs  $\text{SiO}_2$  31.1 per cent.,  $\text{FeO}$  37.5 per cent.,  $\text{MnO}$  1.5 per cent.,  $\text{CaO}$  14.1 per cent.,  $\text{ZnO}$  10.0 per cent.,  $\text{Pb}$  0.77 per cent.,  $\text{Ag}$   $1\frac{1}{4}$  oz. per ton; the matte contains  $\text{Cu}$  3.0 per cent.,  $\text{Pb}$  7.8 per cent.,  $\text{Ag}$  21.6 ozs. per ton, and  $\text{Au}$  0.06 oz. per ton; the bullion 87.5 ozs.  $\text{Ag}$  and 1.49 ozs.  $\text{Au}$  per ton. The total recovery of metals is  $\text{Pb}$  92 per cent.,  $\text{Ag}$  90.8 per cent.,  $\text{Au}$  92.4 per cent.

\* *Min. Ind.*, vol. xvi., p. 668.

† *E. and M. J.*, Feb. 17, 1906, p. 318.

## MEXICO.

Several of the large custom smelting plants in this country are equipped quite on the lines of the most modern plants in the United States, and many references have been already made in previous chapters to details of the practice at certain of these works,\* particularly in regard to the methods of handling slag, matte, &c.

**Aguascalientes.**—At this, the largest single-smelting plant in Mexico, some ores are smelted upon a lead basis, but the majority by employing copper matte as a vehicle for collecting the precious metal contents. Some particulars of the lead smelting practice are given in Table XXXIII. There are four side-feed lead furnaces, 42 by 128 inches, with cast-iron jackets having 16 tuyeres and an effective height of charge column of 17 feet, which, for the blast pressure of 40 ozs., is rather low, and consequently much trouble is experienced through hot top. Lead being comparatively scarce, a matte fall of not over 11 to 12 per cent. is allowed, and the bullion produced runs from 265 to 400 ozs. of silver, containing in addition about 0.2 per cent. Cu, 0.4 to 0.5 per cent. As, and 0.7 to 0.8 per cent. Sb. The first matte produced runs from 9 to 15 per cent. copper, and instead of returning it to the lead furnaces, it is usually roasted and sent over to the copper furnaces, whence, however, the lead contents come back in the form of flue-dust. The matte, after crushing to 6-mesh, is roasted in long hand-rabbed reverberatories fired by Taylor gas-producers, as at Monterey, using a mixture of coke breeze and Laredo coal. Much flue-dust is made in the lead furnaces, and is collected in a long brick flue, 14 × 17 feet inside and 716 feet long, with a dust-chamber 70 × 32 feet, the whole terminating in a brick chimney. The flue-dust, instead of being bricked in a White press as is the roasted matte and flue-dust from the copper furnaces, is bricked by hand with clay (one-sixth by volume), and burned in kilns at a total inclusive cost of only 1s. 9d. per ton of 1,000 kilos. Double separation is adopted for matte and slag by means of two forehearth in series, the lead bullion is taken by means of dumping pots to a central drossing plant, as described in Chapter x.

**Monterey, No. 3.**—At this plant there are eight side-feed furnaces with overhead downtake, measuring 44 by 144 inches, with two tiers of water jackets, the lower being of cast and the upper of wrought iron; 16 tuyeres supply blast at 2½ to 3 lbs. pressure, and the capacity of each furnace is from 135 to 170 tons of charge per twenty-four hours, the effective height of charge column being 16 feet 5 inches, and the coke consumption 12 per cent. on the charge exclusive of slag. Table XXXIII. gives a good many particulars of the practice at these works, and other data have been already given in regard to roasting (Chap. v.), slag composition (Chap. vii.), separation of slag from matte (Chap. viii.), and handling of bullion (Chap. x.).

**San Luis Potosi.**—At this plant there are ten furnaces, formerly top-feed, but modified to side-feed with overhead downtake. They measure 40 inches by 140 inches at tuyeres, with an effective height of 17 feet of charge column, and have cast-iron jackets with seven tuyeres a side. The practice

\* In 1902 the author made a tour of the Mexican plants for the purpose of writing detailed descriptions of them all, but his detailed notes were, unfortunately, lost or mislaid, and such information as is here given is chiefly from memory, supplemented by correspondence. See also Hahn, *Trans. I.M.M.*, vol. viii., p. 277.

at these works in regard to roasting (Chap. v.) and to matte and slag separation (Chap. viii.) has been described in detail. Lead and iron ore being scarce at this plant, and sulphide ores abundant, roasting is an important feature, the matte fall is high, and the concentration furnace is worked with an Orford overflow forehearth, campaigns in which last as much as three months. The concentrated matte is worked up to blister copper by methods described in the *Metallurgy of Silver*. The tonnage put through per furnace is about 75 tons of ore only, in addition to much limestone flux, roasted matte, and slag, the blast pressure being 40 ozs. The slag from the dump is put through a crusher, and removed by the Ry. Co. for use as ballast.

**Mapimi.**—Here a plant of three large new furnaces, 40 inches by 140 inches at tuyeres, has replaced one of six smaller and lower furnaces. The furnaces have cast-iron jackets with seven tuyeres a side and a brick superstructure with side-feed and overhead down comer. The ores treated in this plant are arsenical, and consequently a good deal of speiss is formed, the composition of which, and of the slag, have been given in Chap. x. Reference has also been made in Chap. viii. to the separation of slag and matte. The flue-dust made is highly arsenical (over 30 per cent. of  $\text{As}_2\text{O}_3$ ), but hitherto no use has been made of it, huge mounds of it being tipped in gullies near the smelter and covered with ashes and other refuse merely to get rid of it with a minimum of danger to man and beast.

#### SOUTH AMERICA.

At Casapalca (Peru), 14,000 feet above sea level and 2,000 feet above timber line, rich lead and silver ores were, until recently, smelted in water-jacket furnaces with imported English coke, costing \$40 per ton. The ores consist of—(1) Lump galena with 45 per cent. Pb and 50 ozs. Ag per ton; (2) jig concentrates with 158 ozs. silver per ton; and (3) Vanner concentrates with 209 ozs. silver per ton. All the concentrates contain 6 to 12 per cent. of lead and 3 to 5 per cent. copper. The ores, mixed in the proportion of two parts lump galena to one part concentrates, are charged in 1-ton charges into a single-hearth reverberatory for roasting and slagging. Each charge takes six hours to work, the fuel being 300 lbs. of soft coal (local, costing \$16 per ton) and 250 lbs. of dried llama dung. The slagged ore contains Pb 34·5 per cent., Cu 9·3 per cent., Ag 0·34 per cent.,  $\text{SiO}_2$  17·2 per cent.,  $\text{Fe}_2\text{O}_3$  22·5 per cent., CaO 6 per cent., S 6·5 per cent.—total, 96·34 per cent.

The slagged ore is broken into fist-sized pieces and smelted in a 36-inch round American water jacketed furnace. The average charge is 18,000 lbs. slagged ore, 500 lbs. of ancient slags with 45 per cent. Pb and 2 to 5 ozs. silver per ton, 300 to 500 lbs. of oxidised ores, 1,000 lbs. of bog iron ore, 1,500 lbs. of limestone, and 1,500 lbs. of fowl slags with 8 to 12 per cent. Pb. The products are work-lead with 235 ozs. silver per ton, and copper matte with 27 to 50 per cent. Cu, 7 to 61 per cent. Pb, and 38 to 150 ozs. silver per ton. The slags average  $\text{SiO}_2$  37 per cent.,  $\text{FeO}$  45·2 per cent., CaO 8·6 per cent., Pb 3·4 per cent.; when not containing over 6 per cent. lead they are considered "clean" and are thrown away, as it would not pay to re-smelt them on account of the high cost of fuel. The coke consumption is 14 to 17 per cent. of the charge, which is small enough considering the very high altitude.

Lead having become comparatively scarce at Casapalca, the mixed sulphide ores, together with dry silver ores, are now smelted on a matte basis, as described in the *Metallurgy of Silver*.

At Antofagasta (Chili) \* the average ore mixture from the Pulacayo Mine before roasting contains  $\text{SiO}_2$  20 per cent., Fe 20 per cent., Zn 18 per cent., Pb 7 per cent., Cu 1 per cent., Sb 1 per cent., S 30 per cent., with some Sn and As, and it is smelted in water-jacketed furnaces with removable crucibles on wheels. The jackets are simple plates of cast iron with bent 1-inch wrought-iron pipes cast in for water circulation.

Regulation of the water circulation is of less importance with this type of jacket than with the ordinary water jacket, since the pipes will, upon occasion, even stand being allowed to get red hot through temporary failure of the water supply without sensible damage, unless the burning has lasted too long, or has been repeated frequently. Less water is also required with these than with the ordinary type of jacket. The crucible consists of a cast-iron bottom plate, 2 inches thick, with a raised rim to hold in position the six cast-iron side-plates, which are bolted together, leaving a gap in the centre of each long side; it is lined at first with firebrick on edge, inside which a crust of antimonial speiss soon forms and prevents further leakage.

The charge-column is kept low, only about 8 feet above the tuyeres, and the blast pressure does not exceed 16 ozs. The lead produced is small in quantity and very impure, containing besides 75 per cent. Pb, Sb, Cu, Sn, and Zn in large amounts. With charges carrying 17 to 18 per cent. of Zn, these furnaces run for two or three months before the "hangings" become so large as to cause much overfire, and their removal by barring down is much facilitated by the removable crucible, which can be let down, run out, and replaced by another when the furnace has been cleaned from its accretions.

\* *E. and M. J.*, Sept. 2, 1899, p. 279.

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## CHAPTER XII.

### BLAST-FURNACE LEAD SMELTING—COSTS AND LOSSES— ORE PURCHASING.

**Cost of Smelting.**—The cost of smelting lead ores in the blast furnace varies within wide limits, the principal factors which affect it being—

1. The chemical composition of the predominating available ore supply, its richness in lead, the nature of the gangue, and the proportion of the whole which requires roasting before smelting.

2. The cost of labour at the smelting centre.

3. The cost of fuel and fluxes delivered at the works.

4. The cost of refining locally, or of getting the product to a refining works at a distance.

Taking the case of a so-called “neutral” or self-fluxing ore, in which the percentages of silica and of iron are equal, with sufficient lead (10 per cent.), and not over 4 or 5 per cent. of sulphur, so that it can be smelted without roasting, it is customary at the large smelting works in Colorado and Utah to reckon about \$3.00 per ton as the bare cost of smelting upon a large scale.

**Labour.**—The amount of labour required varies to some extent with the size of the furnace, but to a much greater extent with the general lay-out of the feed- and slag-floors, and with the nature of the arrangements adopted for handling the charge and disposing of the slag. Formerly with hand charge-buggies and slag-pots a furnace of 42 by 140 inches at tuyeres would require per twenty-four hours, besides two foremen, above, three feeders, two weighmen, six to eight charge wheelers; below, two furnace-keepers, two slag-tappers, two helpers, and a variable number of slag pullers and labourers. Regular daily wages in Denver and Pueblo are:—Foremen, 16s. 8d.; furnace-men, feeders, and enginemen, 11s. 6d. to 12s. 6d.; weighmen, 8s. 6d. to 9s. 6d.; charge wheelers, 7s. 6d.; labourers, 6s. 8d. Coke costs about £1 per ton, limestone, with 50 per cent. excess CaO, 6s. 3d. per ton; iron ore, with 70 per cent. excess FeO, 20s. per ton; coal for boilers, 12s. 6d. per ton; slack, 6s. 3d. per ton.

At *Bingham* \* (Utah) six large blast furnaces, 45 by 160 inches, smelting 170 tons each per twenty-four hours, require per shift only the following force—viz., on top, 18 men on the bedding floor, 3 men on the charging cars and feed-floor; and below, 2 motor men and 10 matte and slag men, besides, for each furnace, 1 tapper, 1 helper, and 1 potman per shift. This works out at per furnace per shift  $3\frac{1}{2}$  men on top and 5 men below, or  $25\frac{1}{2}$  men per furnace per twenty-four hours, or by tonnage 6.6 short tons of charge smelted per man per twenty-four hours.

\* *Min. Ind.*, vol. xvi., p. 668.

At *Laurium* \* (Greece) the new large furnaces which put through from 275 to 300 metric tons per day of a self-fluxing charge require only 36 men in all per twenty-four hours—namely, 8 above and 4 below on each 8-hour shift, which works out at 7.6 to 8.4 metric tons of charge smelted per man per twenty-four hours. Since labour is cheap, the total cost for labour and supervision does not exceed 1s. 2½d. per metric ton of material handled, which is, doubtless, the absolute high-water mark of cheapness in smelting.

Hofman gives † the following estimate of the daily cost of smelting in a single furnace, 42 by 140 inches, based upon Colorado data, and assuming the furnace to put through per twenty-four hours 100 tons of charge—viz., 80 tons of ore and 20 tons of flux.

<i>Materials—</i>			£	s.	d.	£	s.	d.
20 tons flux at an average of 8s. 4d. (\$2.00),			8	6	8			
15 tons coke (=15 % on charge) at 20s. 10d. (\$5.00),			15	12	6			
5 tons coal at 12s. 6d. (\$3.00),			3	2	6			
Steel castings, oil, waste, and sundries,			1	5	0			
						28	6	8
<i>Pay Sheet—</i>								
1 superintendent, at per man per day, 41s. 8d. (\$10.00),			2	1	8			
1 assayer, and chemist „ „ 18s. 9d. (\$4.50),			0	18	9			
2 foremen, „ „ 16s. 8d. (\$4.00),			1	13	4			
2 feeders, „ „ 11s. 3d. (\$2.70),			1	2	6			
2 weighers, „ „ 9s. 4½d. (\$2.25),			0	18	9			
8 charge wheelers, „ „ 7s. 6d. (\$1.80),			3	0	0			
2 furnacemen, „ „ 11s. 3d. (\$2.70),			1	2	6			
4 helpers and tappers, „ „ 7s. 6d. (\$1.80),			1	10	0			
5 labourers, „ „ 6s. 8d. (\$1.60),			1	13	4			
1 lead sampler, „ „ 8s. 4d. (\$2.00),			0	8	4			
1 carter, with horse, „ „ 11s. 5½d. (\$2.75),			0	11	5½			
2 engine-drivers, „ „ 12s. 6d. (\$3.00),			1	5	0			
2 stokers, „ „ 8s. 4d. (\$2.00),			0	16	8			
1 blacksmith, „ „ 12s. 6d. (\$3.00),			0	12	6			
						17	14	9½
<i>Add—</i>								
Delays and repairs, 5 % on above, } 10 %.						4	12	1½
Unforeseen, 5 % „								
Cost of smelting 80 tons of ore (plus fluxes) (\$243.26),						50	13	7
Cost of smelting 1 ton of ore (\$3.04),						0	12	8

At a few favoured localities, owing to cheaper labour and fuel, the total cost is lower than this ; while at others, where fuel is dearer, the cost is higher. It should be borne in mind, when statements are made as to exceptionally low costs, that frequently the “cost of smelting” is put down at the bare figures for labour, fuel, and fluxes alone, without any allowance for power required, for general expenses of management, or for repairs, interest, and depreciation.

The above figures refer to the smelting of ores which require no roasting. The cost of roasting at Denver or Pueblo varies from \$1 (4s. 2d.) in automatic furnaces up to from \$1.50 to \$2 (8s. 4d.) in the ordinary hand furnaces, but to these figures something must be added for re-treating flue-dust, and on account of the sulphur left in the ore which forms matte, and so adds to

\* *Private communication*, 1908.

† *Metallurgy of Lead*, 1906, p. 407.

the expense of treatment. At least \$2.50 (10s. 6d.) per ton should be reckoned as the extra cost of handling an ore that requires roasting. The total cost of treatment in the case of a neutral sulphide ore which requires roasting will thus be brought up to a total of \$5.54 (23s. 2d.) at Colorado prices.

*Cost of Smelting in Mexico.*—Hahn \* gives the cost of smelting 183,000 tons of ore during the year 1896 at the No. 3 *Smelter*, Monterey, as \$6.86 Mexican currency, equivalent at 24d. to the dollar to 13s. 9d. per ton of 2,000 lbs., or 15s. 4d. per ton of 2,240 lbs., made up as follows:—Coke \$2.50 (5s.), coal, wood, and charcoal \$0.36 (8½d.), limestone flux \$0.05 (1d.), labour \$1.28 (2s. 6d.), repairs, tools, and supplies \$0.59 (1s. 3d.), taxes \$1.08 (2s. 1d.), general expenses, agency, insurance, &c., \$1.00 (2s.).

The same author gives the cost of smelting at *Aguascalientes* as \$8.55 (17s. 1d.) per ton of 2,000 lbs., of which, however, no less than \$2.61 (5s. 6d.) was for government taxes of various descriptions, including export tax.

TABLE XXXIV.

	Globe, Denver, 1893.	Sm. No. 3, Monterey, Mexico, 1896.	Aguas- calientes, 1896.	Denver, Pueblo, 1906.	Frem- ontle, W.A., 1904.	Works "A," 1903.
Reference,	1	2	3	4	5	6
Wt. of "ton" to which data refer, . . . . .	2000 lbs. s. d.	2000 lbs. s. d.	2000 lbs. s. d.	2000 lbs. s. d.	2240 lbs. s. d.	1000 ks. s. d.
General labour and super- vision, . . . . .	4 9½	2 6	..	6 11	1 3	1 2 40
Furnace labour, . . . .					5 6	
Assaying, sampling, weighing, &c., . . . . .					2 2	
Power charges, . . . . .	0 2½	0 8½	..	3 6	1 0	0 7 68
Fluxes (chiefly limestone), .	0 11½	0 1	..		8 4	..
Coke (and other furnace fuel),	5 8	5 0	..		13 11	5 6 70
Handling slag, . . . . .	1 2½	1 3	..	0 8	0 4	0 2 86
Electric lighting, . . . .						0 0 96
General maintenance, repairs, and sundry materials, .						0 4 80
General expenses, export and other taxes, . . . . .	12 10	9 6½	12 4½	11 1	32 6	8 1 42
Interest on capital, . . . .	0 10	4 2	5 5½	..	..	2 4 80
	0 6	..	..	..	..	..
	14 2	13 8½	17 10	..	..	10 6 22

*References.*—1. Iles, *E. and M. J.*, Sept. 16, 1899. 2. Hahn, *Trans. I.M.M.*, 1900, vol. viii., p. 277. 3. *Ibid.* 4. Ingalls, *Trans. A.I.M.E.*, 1906, vol. xxxvii., p. 635, 5. *Private communication*, L. Pitblado. 6. *Private communication*.

At both of these works the coke is imported from the United States (native coke being of poor quality) and costs from \$12.60 to \$14.00 (25s. 3d. to 28s.) per ton of 2,000 lbs., while, although American foremen are required, the bulk of the labour is native, and exceedingly cheap, the wages paid

\* *Trans. I.M.M.*, vol. viii., p. 277.

ranging from 37 centimes (9d.) up to \$1.75 (3s.) per day for the more important positions.

Smelting works are always chary about publishing their working costs, and, therefore, it is not possible to give here detailed figures referring to the most modern practice at large and well-equipped custom smelting works like those of the American Smelting and Refining Coy. The preceding Table XXXIV., however, gives a few actual figures of cost which, while they confirm in general the above figure of 12s. 8d., quoted as being an average under Colorado conditions, show at the same time how the costs are affected by varying prices of labour and of fuel and fluxes.

In an interesting recent article Ingalls\* discusses the cost of smelting under the conditions prevailing to-day at the great custom plants in Colorado and Utah. The average cost of smelting per ton of charge smelted in nine great works is given as \$3.607.

About 80 per cent. of the charge being ore, this works out at about \$4.50 per ton of ore smelted. These figures are checked in another way, the roasting cost being estimated at \$2 per ton, and it being estimated that for each ton of charge smelted about 0.4 ton requires roasting. We have then—

Furnace cost per ton of charge—	
Labour, power, and supplies, . . . . .	\$1.66
Coke, . . . . .	0.84
Administration, . . . . .	0.16
	<hr/>
	\$2.66
Ore being 80 per cent. of charge, equals per ton ore, . . . . .	\$3.33
Per ton charge smelted about 0.4 ton roasted; 0.4 tons at \$2, . . . . .	0.80
	<hr/>
Per ton of ore handled, . . . . .	\$4.12
	<hr/>

which fairly confirms the former figure of \$1.50 derived from the published reports of the American Smelting and Refining Co., which figure, however, it will be understood merely represents the bare operating cost. To it must be added the following items:—

Interest and amortisation. The cost of works for treating 1,000 tons of ore per day may be taken at about \$1,000,000 = \$3 per ton per annum. At 10 per cent. amortisation and 6 per cent. interest, this amounts to per ton of ore treated, . . . . .		\$0.48
Interest on stocks of ore. About one month's supply is kept on hand, say 30,000 tons, at an average value of, say, \$30 per ton. At 6 per cent. the tonnage charge for interest amounts to . . . . .		0.15
And interest on product during the time taken for realisation, say one month, is . . . . .		0.15
Add the cost per ton of ore handled taken from the average published figures, . . . . .		4.50
	<hr/>	
Total smelting cost, . . . . .		\$5.28
	<hr/>	

**Losses.**—The losses in smelting are—(1) In slag thrown away; (2) in flue-dust and fume incompletely recovered; (3) in bye-products, matte, speiss, flue-dust, &c., which suffer further treatment losses before they can be brought into a marketable condition.

\* *Min. Ind.*, vol. xvii., pp. 608-617.

When producing a lead bullion of about 300 ozs. Ag per ton, perfectly clean well-proportioned slags will run as low as  $\frac{3}{4}$  per cent. lead by dry assay and  $\frac{1}{2}$  oz. silver to the ton; the general average, however, in good work is probably not much under 1 per cent. lead and  $\frac{3}{4}$  oz. silver, while carelessness in making up the charges or neglect on the part of the furnace foreman may easily result in the production of slags which separate imperfectly from matte and contain 2 per cent. lead and  $1\frac{1}{2}$  ozs. of silver; especially if the ore be somewhat rich in lead, and the furnace one of the old type with comparatively low column of charge and pressure of blast.

With a smaller production of bullion running up to 400 ozs. to the ton the loss in slag is proportionately greater, and may average in good work from 1 per cent. or more of lead and  $1\frac{1}{4}$  to  $1\frac{1}{2}$  ozs. of silver per ton.

In some cases, where fuel and flux are dear (as, for example, at Broken Hill, N.S.W.), or where the ore charge is rich in zinc, it may even be better policy to throw away slags containing 3 or 4 per cent. lead rather than to attempt to clean them by a further expensive consumption of fuel, and the addition of barren fluxes, with the attendant reduction of furnace capacity to which these give rise. The average percentage loss of silver and lead in slags on a charge containing 10 per cent. lead and 30 ozs. silver may be reckoned at not much less than 10 per cent. of the lead and 2 per cent. of the silver as calculated by the results of the commercial uncorrected dry assays, which are, as has been shown, far below the truth.

With regard to slag losses, it is desirable to bear in mind that losses which would be intolerable at Swansea, at Freiberg, or at Denver, may actually indicate the best possible practice in remote parts of Mexico or South America. The object of the metallurgist should be to make the largest possible profit per ton of ore treated, and if it is more profitable to produce a slag with 5 per cent. lead and 4 ozs. silver per ton than to flux the charge in such a way as to make a cleaner slag, as is the case to-day in parts of Central and South America, then the foul slag is the proper slag to make. The object of metallurgy being to make money, the best metallurgical practice is not that which shows the most scientific perfection, but that which makes the most profit.

**Slag Losses.**—We have seen (Chap. x.) that slag losses depend on the following factors:—

1. The composition of the slag.
2. The bullion-fall.
3. The matte-fall.
4. Proper separation of the matte.
5. The richness or poverty of the lead.

We have also seen that the losses of lead and silver in slag may range, even in good work with correctly compounded slags, from  $\frac{1}{2}$  per cent. Pb and  $\frac{1}{2}$  oz. Ag with a yield of work-lead of 14 per cent. or over and a bullion of about 200 ozs. silver, up to  $1\frac{1}{4}$  per cent. Pb and  $1\frac{1}{2}$  ozs. Ag, when the yield of work-lead is only 10 or 11 per cent., and its silver contents run up to 400 ozs. to the ton.

Generally speaking, silver and lead losses are more or less concurrent, inasmuch as it is impossible to lose a large amount of lead in the slag without at the same time losing silver, but the losses of the two metals do not run entirely on parallel lines, nor are they affected equally by the same factors. It will be best, therefore, to consider the loss of each metal separately.

*Loss of Lead in Blast-furnace Slags.*—The lead contents of slag may vary, as has been seen, from about  $\frac{1}{2}$  per cent., or even less in exceptional cases of favourable composition, high matte fall, and good separation, up to  $2\frac{1}{2}$  or 3 per cent. with viscous siliceous-aluminous slags, low matte fall, and poor separation of a light zincy matte with little or no copper. It does not increase with increased bullion fall, but rather decreases, and for all ordinary conditions of custom smelting on a lead basis may be reckoned at an average of from 1 to 1.2 per cent.

The loss of lead is partly chemical and partly mechanical, and is largely dependent upon the chemical composition of the slag. Thus, comparing the slags numbered 19 and 21 in Table XVII., the former contains an average of barely 1 per cent. Pb, whereas the latter averaged  $1\frac{3}{4}$  per cent., and often in practice runs up to 2 per cent. Pb. Taking into account that both slags are accompanied by a bullion fall of about 12 per cent., and that both charges are low in sulphur (the matte fall being in the one case 2 per cent. of a matte with 15 per cent. Cu, and in the other  $4\frac{1}{2}$  per cent. of a leady and zincy matte almost free from Cu), it is clear that most of the excess lead in the second slag is in the combined form, although it is true that the greater viscosity of the aluminous slag (No. 21) renders necessary much better settling facilities in order to approach the same freedom from shots. As the bullion fall decreases, this aluminous slag becomes more and more leady, thus with a bullion fall of only 7 per cent. the slag produced contained no less than  $2\frac{1}{2}$  per cent. Pb, chiefly, no doubt, in the form of silicate.

Other things being equal, the farther a slag departs from the monosilicate type the greater the tendency for lead to be found in it combined as silicate. The Broken Hill slags of former days, which were very similar in composition to No. 21, ran even higher in lead (average 3 per cent.), but for this, besides the high silica, the low matte fall, and the absence of copper, there was an additional reason in the presence of nearly 2 per cent. of alkalis, which always increase the tendency of lead to become combined in slag, partly, no doubt, because of the lowering of the formation temperature of the slag to which they give rise.

Increasing the lime in slags always decreases the loss of lead, doubtless because of the increased formation temperature and decreased specific gravity, the former of which tends to improve reduction, while the latter improves separation.

Supposing, however, the slag to be correctly proportioned so as to have a not too low formation temperature, and to be quite fluid when molten, then the loss of lead in the slag is almost entirely a matter of reduction; cutting down the fuel will always increase the loss, while raising the percentage of fuel will reduce it.

The *loss of silver in blast-furnace slags* is entirely mechanical, and is due almost entirely to minute particles of matte and of bullion carried in suspension owing to imperfect settling. It is, therefore, dependent rather upon the degree of fluidity possessed by the slag, and upon the bullion and matte fall, than upon the exact composition of the slag. The actual loss varies, even with correctly proportioned slags, from  $\frac{1}{2}$  oz. up to  $1\frac{1}{2}$  ozs. per ton, and may be much higher when the percentages of alumina and of zinc are high. Granting that the loss is in the form of suspended globules or particles, it is

evident that it will be increased by all those causes which affect the separation of matte from slag, namely :—

1. By increased viscosity of slag, from whatever cause arising.
2. By increased specific gravity of slag, due to excessive iron, baryta, or zinc contents.
3. By decreased specific gravity of matte, due to zinc, in cases when a good deal of zinc sulphide exists in the ore charge.

Seeing, moreover, that the silver contents of ordinary lead mattes are directly proportional to the combined percentages of lead and copper, each of which brings in silver in the proportion in which that metal is contained in the lead bullion produced, it will be obvious that the silver contents of the slag must also be in some degree proportional to the silver contents of the bullion, and directly proportional to the lead contents of the matte. Imperfect reduction of lead, therefore, followed by a rise in the lead contents of the matte, is at once reflected in the silver assay of the slag.

The loss of silver in slag must also depend to some extent upon the mode of combination of the silver in the original ore charge, for it will be readily understood that the loss must be higher when a large part of the silver contents of the ore exists as chloride, sulphide, or other high-grade silver mineral in siliceous or other "dry" ore free from lead, disseminated, therefore, in excessively minute particles, which have to be collected or dissolved by means of globules of lead reduced from lead carbonate or roasted galena ore, forming a separate item in the ore charge, than when most of the silver in the charge is already combined with lead as argentiferous galena, tetrahedrite, or other sulphide mineral. In all cases where rich "dry" silver ores form an important part of the charge, it is found that the slags show a tendency to run up in silver contents irregularly, more particularly when the matte fall is low; and it is probable that in such cases, besides the ordinary and more or less uniform loss in particles of matte which are carried in suspension by the slag and escape settling, there is a small variable loss in particles of metallic silver or of rich sulphide carried in suspension in the slag, which have escaped collection in the work-lead and matte.

Slag produced concurrently with a high matte fall is not likely *ceteris paribus* to be more free from silver than a slag of the same composition accompanying a low matte fall, except in so far as a low matte fall has a less potential capacity for collecting minute shots of precious metal or of lead suspended in the slag, and that the "mass-effect" of a large matte fall may perhaps assist to some extent the segregation and coalescence of shots and their separation from the slag.

The composition of the matte, too, apart from the percentages of lead and zinc in it, has a notable effect on the silver contents of the slag, for it has been observed that when normally producing mattes low in copper, the addition of a little oxidised copper ore to the ore charge has an immediate effect in lowering the silver assay of the slag. This can hardly be accounted for by any fractional increase in the specific gravity of the matte; it may, however, perhaps be due to the greater specific affinity for silver sulphide possessed by  $\text{Cu}_2\text{S}$  as compared with  $\text{PbS}$ .\*

\* As indicated by the perfection of the solid solutions which  $\text{Ag}_2\text{S}$  and  $\text{Cu}_2\text{S}$  form with each other in all proportions, see the papers by Friedrich and Fulton and Goodner, already quoted in Chapter x. in the remarks on lead mattes.

The loss of silver in slag appears to be increased by the predominance of lime as a slag base, in spite of the fact that an increase in lime decreases the specific gravity of a slag, and so improves the separation of matte, while, conversely, the silver loss is diminished by the substitution of iron for lime.\* No satisfactory explanation is at present forthcoming to account for this, but it has been already noted that excess of lime tends to increase the volatilisation of silver in blast-roasting.

Before considering the *loss of gold*, it will be well to refer to the following Table XXXV., which gives a few data bearing upon losses of both silver and gold in slags. It will be noticed that, in addition to the silver and gold assays of the mattes, the precious metal contents of these have been calculated back on to their lead and copper contents taken together, and that the ratio between the silver contents of the work-lead and that of the combined lead and copper contents of the matte varies only between the comparatively narrow limits of 1:0.6 and 1:1.55, in spite of very wide variations in the bullion fall and matte fall, and of the fact that the extreme figures refer to cases where the bullion is of low grade and the matte poor in copper. Taking columns 3, 5, 6, and 8 of the table, which refer to what may be considered the normal condition of affairs in custom smelting, with a bullion fall of  $8\frac{1}{2}$  to 12 per cent. of work-lead carrying 90 to 266 ozs. and a matte carrying from 4 to 15 per cent. of copper, it will be seen that the variation in the ratio referred to is only from 1:0.86 up to 1:1.11, in spite of considerable variations in the bullion and matte fall. This confirms what has been stated already—namely, that the affinity of metallic lead for silver is no greater than that of  $\text{Cu}_2\text{S}$ , or of a mixture of  $\text{Cu}_2\text{S}$  and  $\text{PbS}$ ; and that, so far as can be seen, both of these sulphides, and more particularly the former, carry into the matte as much silver as is contained in the work-lead itself.

The gold contents of the matte, on the other hand, bear no distinguishable relation to the gold contents of the bullion produced at the same time, for the ratio between the gold assay of the bullion and the gold assay of the matte calculated back upon its lead and copper contents taken together varies between 1:3.7 and 1:28.6. This seems to indicate that so far from the affinity for gold of lead and copper sulphides being at all comparable with that of the metals themselves, as is the case with silver, it is very much less. The efficiency of matte, therefore, as a vehicle or medium for concentrating gold is, compared with lead bullion, much lower than its efficiency for concentrating silver.

It is difficult to draw any inference from reported slag assays, because so much depends upon the degree of settling which the slags have undergone, but, taking the figures given in the table, it will be seen that the silver contents of slag vary somewhat approximately with the silver contents of the bullion and of the matte produced at the same time; the ratio varying in the case of the bullion from 1:65 up to 1:317, or, leaving out the lowest

\* According to a communication from L. Pitblado to the author, experiments made at the Fremantle Smelting Works showed that a high percentage of lime in the slag actually increased the gold as well as the silver losses. The author is unable to accept this result as of general application, and is inclined to think that it can only be true in cases where lime has been added in excess of the combining power of the silica, so as to form spinel in the slag, which would increase its viscosity, and so prevent the settling out and coalescing of minute suspended particles.



TABLE XXXV.—DATA OF RECOVERIES AND SLAG LOSSES.

Reference.	Col. S. & R. Co., Pueblo, Colo.		Globe Works, Denver, Colo.	Laurium, Greece.	Monterey, Mexico.	Mapimi, Mexico.	Cockle Creek, N.S.W.	Dapto, N.S.W.	Fre- mantle, W.A.
	Average, 9 mos.	Average, 6 mos.							
Average bullion fall,	..	..	12.5%	11%	10%	8.4%	..	12	10
" matte-fall,	10%	10%	9%	1-1½%	2%	2%	..	12	4.5
" composition of matte,	..	..	11	7.5	14	18	..	15	16.5
" " " "	..	..	6	..	..	5	..	..	..
" " " "	..	..	15	2	15	10	..	4	..
" composition of speiss,	..	..	..	0.5	2	5	..	..	..
Wt. of ore : wt. of slag :: 100 :	..	..	95	..	95	90	..	125	132
<i>Silver—</i>									
Lead, Assay of, Ag ozs. per ton,	..	..	266	49	249.1	239.5	71.5	90	65
Matte " " "	..	..	77	8	65	76	..	20	7
Speiss, " " "	..	..	24.7	3.5	20	39	..	..	..
Slag, " " "	..	..	0.84	0.33	1.0	1.6	1.1	0.75	0.43
Ag assay of matte calculated on to Pb+Cu contents,	..	..	296	81	224	271	..	105	42
Ag assay work-lead : Ag assay Pb+Cu in matte :: 1 :	..	..	0.90	0.60	1.11	0.88	..	0.86	1.55
Ag assay slag : Ag assay work-lead	..	..	317	148	249	148	65	120	150
Ag assay slag : Ag assay matte	..	..	91	25	65	47.5	27	14	27
Proportion of total Ag contents of charge in work-lead,	..	..	80.33	..	93.03	82.56	..	75.63	89.48
" matte,	..	..	16.74	..	2.42	6.24	..	16.81	4.28
" speiss,	..	..	..	..	..	4.39	..	..	..
" slag,	..	..	1.93	..	3.55	5.91	..	6.58	6.25
<i>Gold—</i>									
Work-lead, Assay of, Au ozs. per ton,	3.73	5.67	3.49	0.13	1.67	2.35	1.95	30.00	25.00
Matte, " " "	0.2	0.2	0.08	tr.	0.015	0.093	0.030	0.20	0.28
Speiss, " " "	..	..	0.48	0.016	..	0.197	..	..	..
Slag, " " "	0.0034	0.00473	0.004	..	..	0.00232	0.0011	0.02	0.04
Au assay of matte calculated on to Pb+Cu contents,	1.0 oz.	1.0 oz.	0.31	..	0.062	0.033	..	1.05	1.70
Au assay work-lead : Au assay Pb+Cu in matte :: 1 :	3.73	5.67	11.3	..	3.21	7.12	..	28.6	14.7
Au assay slag : Au assay work-lead	1.097	1.199	872	..	682	1010	1770	1500	625
Au assay slag : Au assay matte	57	42	20	..	6.1	40	27	10	7
Proportion of total Au contents of charge in work-lead,	..	..	96.46	..	97.60	94.6	..	97.59	96.53
" matte,	..	..	1.55	..	0.09	0.91	..	0.64	0.50
" speiss,	..	..	..	..	..	2.54	..	..	..
" slag,	..	..	0.89	..	1.40	1.00	..	0.68	2.01

References.—1 and 2. *Private communication*, A. Ehlers. 3. *Iles, Metallurgy of Lead*. 4. *Private notes*, 1904. 5, 6, and 7. *Private communications*, 1904. 8. *Private communication*, E. S. Weinberg. 9. *Ibid.*, L. Pitblado.

figure, obtained when working with slags rich in zinc and consequently difficult to separate perfectly from matte, from 1 : 120 up to 1 : 317.

It would appear that the gold contents of slags also bear a somewhat close relation to the gold contents of the bullion produced, for, in spite of the circumstance that the gold assays of the bullion produced vary from 1·67 ozs. all the way up to 30 ozs. per ton, the extreme variations in the ratio between slag and bullion assays for gold are only from 1 : 625 to 1 : 1770. The high slag assays for gold shown in the table are from the Australian works producing from ores rich in gold a bullion low in silver by the use of Broken Hill concentrates as lead-bearing material; it will be remembered that, as already pointed out, the works in question make slags which are not only from this cause high in zinc, but are also, from the nature of the ores treated, high in alumina also, and this means that they are pasty and do not allow of ready separation of matte, more especially as the latter is inclined to be zincy. It seems, therefore, probable that, with well-proportioned slags, the gold loss does not increase at all proportionately to increased gold contents of the bullion. This view is confirmed by Dr. A. Eilers,\* who has kindly supplied the figures given in the first two columns of Table XXXV., each of which represents the average of many months' work at one of the large plants of the Am. Sm. and Ref. Co., and, in addition, quotes a case in which, during a period of six days, rich gold ores were treated in a separate furnace with appropriate admixture of other ores, and yielded a bullion containing 20 ozs. gold per ton with a slag containing only 0·01 oz. per ton, or a ratio of 1 : 2,000.

Beyond this point, however, the gold loss in slags appears to increase proportionately to the gold contents of the bullion, and even somewhat more rapidly. The following figures have been supplied from the results of smelting rich West Australian gold ores at Dapto by Mr. E. A. Weinberg.†

TABLE XXXVI.—SHOWING GOLD LOSSES IN SLAG WITH VARIOUS GRADES OF RICH BULLION.

Bullion. Gold Assay per Ton.	Slag. Gold Assay per Ton.	Percentage Loss of Gold In Slag, calculated on Basis of Slag Tonnage Made.	Ratio, Slag Assay : Bullion Assay.
5 ozs.	0·003	0·42 %	1 : 1666
10 ozs.	0·005	0·35 %	1 : 2000
10-20 ozs.	0·005 - 0·01	0·35 %	1 : 2000
20-30 ozs.	0·01 - 0·02	0·35 - 0·46 %	1 : 1666
30-40 ozs.	0·02 - 0·03	0·46 - 0·54 %	1 : 1400
40-50 ozs.	0·03 - 0·04	0·54 - 0·56 %	1 : 1285
50-60 ozs.	0·04 - 0·05	0·56 - 0·58 %	1 : 1222
60-70 ozs.	0·05 - 0·06	0·58 - 0·60 %	1 : 1182
70-80 ozs.	0·06 - 0·07	0·60 - 0·61 %	1 : 1154

\* In private communication to the writer, dated July, 1904.

† Mr. Weinberg remarks :—"The gold contents of the bullion were kept when possible at about 30 ozs. per ton, but some days, under exceptional pressure and scarcity of lead, the bullion was run up to 60 and 80 ozs. per ton."

Whenever the slags contained above 0.025 oz. per ton, they were saved for re-smelting. It is remarkable that throughout the whole period during which this rich bullion was being produced the matte produced rarely carried more than 0.25 oz. per ton, the matte fall being kept at an average of about 12 per cent. on the charge.

These figures confirm the opinion long held by most metallurgists \* who have considered the matter—namely, that gold in slags exists chiefly, if not exclusively, in the form of excessively minute metallic particles which have not come into contact with the globules of lead reduced in the furnace, and so have not been collected, while their minuteness precludes all possibility of their settling out independently. In connection with this matter, it may be well to remark that collection of these minute particles is effected far more perfectly in the blast furnace than in the process of assaying (as might indeed be expected), since slags produced in assaying such rich ores, whether in the crucible or by scorification, however carefully the assay may be performed, are many times richer in gold than the slags produced in the blast furnace. It is partly for this reason that most smelting works are able to show an overplus of gold as compared with the quantity paid for.

**Losses in Fume.**—The losses in uncondensed fume are doubtless much greater than is generally supposed, owing to the fact (referred to at greater length in the chapter on “Assaying and Analytical Methods”) that the ordinary dry assay for lead and silver give results which are below the truth. In the case of ordinary mixtures of ores averaging, say, 30 ozs., the commercial uncorrected assay for silver is low to the extent, on the average, of 5 or 6 per cent. of the total; while the dry assay for lead on an ore mixture containing 12 per cent. of that metal is low by probably 7 to 10 per cent. of the total present. The real loss in uncondensed flue-dust is, therefore, greater than it is usually reckoned to be.

The loss of *lead* in fume is extremely variable, and depends upon a number of factors; anything which gives rise to over-fire naturally increases the loss of lead in fume. Generally speaking, it may be said that the quicker the rate of smelting the less the fume loss.

The loss of *silver* in fume is always comparatively low, but is increased by the occurrence of chlorides in the charge, whether existing as silver chloride or as chlorides of base metals. Generally, however, it may be said that the loss of silver in true fume is quite low and almost constant—that is, it does not vary directly with the assay value of the ore charge; the loss in dust, however, is, of course, proportionate to the original silver contents of the fine ores blown over.

**Recovery of Metals, Lead and Silver.**—It will be readily understood that for a given set of conditions the loss of lead per ton of ore will be very nearly constant, so that the percentage recovered varies principally with the average lead contents of the charge smelted.

According to Hofman,† the total apparent loss of lead in good work is 6 to 7 per cent., though it not uncommonly reaches twice this amount (corresponding with real total losses of 14 and 21 per cent. respectively). As regards the total apparent loss of silver, Hofman says it “rarely reaches 5 per cent.,”

\* Dr. Eilers, in the private communication above referred to, says:—“I have come to the conclusion that the gold contents of the slag is entirely owing to the exceedingly small flakes of metallic gold or tellurides which could not settle through the slag.”

† *Metallurgy of Lead*, 1906, p. 407.

which would indeed be quite an outside figure. That the real loss of silver is, however, much greater than that shown by assay, is proved by the fact that at some establishments the silver produced is more than that shown by the assays which have served as a basis for purchase. This, however, does not so much indicate perfection in the smelting as imperfection or bias in the assaying. Such gains, in fact, are always due, as pointed out by Hahn,\* “rather to commercial than to technical skill.”

As regards the loss of lead in uncondensed fume, which is so much greater than the corresponding loss of silver, a large proportion could be saved by filtering appliances, but in many places it would hardly pay to use them.

Some data with regard to the proportions of the total precious metal contents of the charge which find their way into each of the furnace products are given in Table XXXV., and the following Table XXXVII. gives similar data from two European works for comparison with more modern practice in America:—

TABLE XXXVII.—APPARENT PERCENTAGE YIELD OF SILVER AND LEAD.

	Pontgibaud, 1874.		Clausthal, 1887.		El Paso, 1892.		Average Good Work, American and Australian Works, 1897-1908.	
	1.		2.		3.		4.	
	Lead.	Silver.	Lead.	Silver.	Lead.	Silver.	Lead.	Silver.
Grade of ore treated, .	40 %	96 ozs.	50 %	23 ozs.	13 %	40 ozs.	11-14 %	20-30 oz.
Percentage yield:—								
In work-lead, . . .	80.04	102.28	80.92	87.01	81.2	86.9	..	..
In foul slag, . . .	2.04	0.66						
In flue-dust, . . .	3.67	0.56						
In matte, . . .	..	..	10.14	13.22	5.0	2.6	..	..
Total yield, . . .	85.75	103.5	91.06	100.23	90.0	97.0	90-95%	97-98%
Loss or gain on com- mercial assays, . .	—	+	—	+	—	—	—	—
	14.25	3.5	8.94	0.23	10.0	3.0	5-10%	2-3%

References.—1. Phillips, *Elements of Metallurgy*, 1874, p. 563. 2. Schnabel, *Handbuch der Metallhüttenkunde*, vol. i., p. 407. 3 and 4. *Private notes*.

Keen competition among smelters in America has led to greater care in assaying, American assayers commonly cupelling at a lower temperature than their *confrères* in Europe connected with smelting plants, with the object of partially compensating for the cupel loss, and so getting a nearer approximation to the truth. It is to this cause that the apparently higher loss of silver in American works should undoubtedly be attributed.

Even the apparent recovery of 98 per cent. of the silver shown by well-managed modern plants is much in excess of the figure which would be shown if every assay were corrected for slag and cupel losses, probably in that case the recovery shown would not be above 95 per cent. at the outside.

*Recovery of Gold.*—The recovery of gold, as shown by the books, charging up only the amount paid for (ores carrying less than  $\frac{1}{10}$  or  $\frac{1}{20}$  oz. per ton not being considered) may range between 98 and 107 per cent. of the amount

\* *Trans. I.M.M.*, vol. viii., p. 278.

shown by commercial assays. The lower figure is only shown in the case of Australian works treating high-grade telluride ores, with nothing but Broken Hill concentrates as lead-bearing material, and insufficient arrangements for flue-dust condensation. In all ordinary cases of custom smelting producing bullion of moderate grade the recovery of gold, according to the most reliable data accessible to the author, varies from 101 per cent. up to 107 per cent. of the amounts shown by the commercial assay, and will average perhaps 102 to 103 per cent.

The surplus in gold is partly to be accounted for by the recovery of that metal from ores in which it existed in quantities too small to be paid for, and also from iron fluxes which frequently contain traces of the metal. Most of the surplus, however, is derived from the fact that the furnace work is more perfect than the assaying, the loss in furnace slags being as was stated above, less than in those produced by the assayer.

The recovery of lead has been much improved since as a result of the adoption of higher furnaces and higher blast pressures, the time during which lead is exposed to the heat of the smelting zone has been much lessened; while concurrently the higher smelting column exerts an important influence in re-condensing lead volatilised in the smelting zone, and in capturing and filtering out great part of the fine particles which, with a lower smelting column, are often blown away as dust. With a good bullion fall of, say, 14 or 15 per cent. and allowing for recovery of lead from matte and flue-dust, it is quite possible to recover 95 per cent. of the value of lead in the charge as shown by dry assay, even without a bag-house; with a bag-house this percentage recovery can be exceeded with even a lower bullion fall. In these figures, however, no account would be taken of small quantities of lead contained in some of the ores smelted, and not paid for, which would make the furnace work appear more perfect than it is in reality.

**Smelting Charges.**—The cost of smelting any given ore is largely influenced by its chemical composition, and is based upon the cost of smelting a self-fluxing ore which needs no roasting, and contains no objectionable ingredients in quantities greater than certain arbitrarily fixed limits, varying with each locality, and indeed from time to time in the same locality. A so-called "neutral" ore is always one in which the insoluble residue does not exceed the percentages of Fe and Mn (not their oxides) added together, but, in addition, the percentages of Zn, S, As, Ba, and other objectionable ingredients must not exceed the limits fixed (say, 8, 5, 1, and 10 per cent. respectively). Beyond these limits special charges are made; for instance, under Colorado conditions the following amounts are often charged and credited:—

	Debit. Cents.	Credit. Cents.
Silica (insoluble), for each unit in excess of iron, . . . . .	10	..
Iron and manganese, each in excess of insoluble, . . . . .	..	10
Lime, per unit, . . . . .	..	6 to 10
Magnesia and baryta, when under 10 per cent., . . . . .	..	as lime
Zinc, per unit in excess of 10 per cent., . . . . .	50	..
Barium sulphate, in excess of 10 per cent., . . . . .	5	..
Sulphur, per unit, in excess of 5 per cent., unless a charge of \$2 be made for roasting, . . . . .	15	..
Arsenic, per unit, . . . . .	variable	..
Lead deficiency below 10 per cent. in the case of dry ores, . . . . .	10	..

Two examples may be quoted of the manner in which the treatment charge is ascertained in accordance with the above scale of charges.

*Dry Ore*—Analysis—Silica 50 per cent., Fe 9 per cent., CaO 10 per cent., BaO 6 per cent., Zn 5 per cent., Pb 4 per cent.

	Debit. Dollars.	Credit. Dollars.
Silica, 50 - 9 = 41 units excess, at 10 cents, . . .	4.10	..
Lime, 10 per cent. } 16 units at 10 cents, . . .	..	1.60
Baryta, 6 " }	..	..
Zinc, no charge, . . . . .	..	..
Normal smelting charge, . . . . .	3.00	..
Lead, "fluxing", 10 - 4 = 6 units at 10 cents, . . .	0.60	..
Nett bare cost of smelting, . . . . .	..	6.10
	<hr/> 7.70	<hr/> 7.70

*Sulphide Lead Ores*.—SiO<sub>2</sub> 20 per cent., Fe 32 per cent., Pb 15 per cent., Zn 15 per cent.

	Debit. Dollars.	Credit. Dollars.
Iron excess, 32 - 20 = 12 units at 10 cents, . . .	..	1.20
Zinc excess, 15 - 10 = 5 units at 50 cents, . . .	2.50	..
Roasting charge, . . . . .	2.00	..
Normal smelting charge, . . . . .	3.00	..
Nett bare cost of treatment, . . . . .	..	6.30
	<hr/> 7.50	<hr/> 7.50

To the above figures which represent the nett bare cost to the works must, of course, be added whatever profit the smelter wishes to make from each individual ore, and this again depends upon its desirability for his purpose, abundance or scarcity, &c., also upon its richness in silver, since the richer the ore the greater the interest on the capital locked up until its metallic contents can be realised, say, on the average, a month or more. Lead ores being usually scarcer than "dry" ores, the treatment charge on them is placed as low as possible; often, in fact, they are treated at bare cost with no margin for profit, while the margin on the dry ores is correspondingly increased.

Some ores in special demand are purchased by the smelting works on terms much more favourable to the producer. Thus, the gold bearing pyrites concentrates of Gilpin County, Colorado, containing under 10 per cent. SiO<sub>2</sub>, were for a long time eagerly purchased by Denver smelters at a smelting rate of \$2.25 per ton, which barely covered the cost of roasting, full value being paid for the gold and silver contents.

**Purchasing Ores.**—The usual prices prevailing in Western America vary from time to time, but are somewhat as follows:—

*Gold.*—Pay for 95 per cent. of the full Mint value, \$20.67 per oz.; or, what amounts to nearly the same thing, pay at the rate of \$19.50 per oz. for all the gold contents, provided not less than  $\frac{1}{20}$  oz.

*Silver.*—Pay for 95 per cent. of New York quotation on day of settlement.

*Copper.*—Pay for all in excess of 2 per cent. by dry assay at about one-half of its gross value, often at \$1 per unit.

*Lead*.—The assumed standard for purchasing lead is a New York price of 4 cents per lb., or \$4 per 100 lbs. At this price the *gross* value of each unit is 80 cents, from which has to be deducted the loss in treatment (at least 10 per cent. of the lead), also the freight to Eastern markets, and the refining charges. In practice, for 50 per cent. lead ores the standard fixed at the basal New York quotation of 4 cents per lb. for lead is from 50 to 55 cents per unit. It is obvious that the losses and charges will fall much more heavily on the poorer ores, hence the following sliding scale is adopted :—

Under 5 per cent. lead		at nothing.	
5 per cent. and under 10 per cent.		at 25 cents per unit of 20 lbs.	
10	"	20	"
20	"	30	"
30	"	40	"
40	"	50	"
50	" and over	55	"

Fluctuations in the New York price of lead are provided for by making an allowance of 1 cent per unit up or down for every 5 cents per 100 lbs. variation in the New York price up or down.

For rich lead concentrates carrying 65 per cent. lead or upwards, and more especially for high-grade ferruginous carbonates, special rates are paid ; and, indeed, such ores are often purchased at their full value, allowing for treatment losses and refining charges, and smelted for nothing, the loss in handling them being recouped by extra charges on the "dry" and less desirable ores generally.

Another scale in use at Colorado smelting plants since 1906 for purchasing lead (copper, gold, and silver being paid for as above explained) is as follows :—

Lead Contents.		Prices and Treatment Charges on Neutral Basis.	
5 to 10 %.	. . . .	25 cents per unit, less	\$8.00 working charge.
10 to 15 %.	. . . .	25	" \$7.00 "
15 to 20 %.	. . . .	25	" \$5.00 "
20 to 25 %.	. . . .	25	" \$4.00 "
25 to 30 %.	. . . .	30	" \$4.00 "
30 to 35 %.	. . . .	30	" \$3.00 "
35 to 40 %.	. . . .	30	" \$2.50 "
40 to 45 %.	. . . .	32	" \$2.00 "
45 to 50 %.	. . . .	35	" \$2.00 "
Over 50 %.	. . . .	40	" \$2.00 "

At *Utah* smelting plants the following scales of payment are at present in force :—\*

*Gold*.—All above 0.03 oz. per ton, at \$19.50 per oz.

*Silver*.—Ninety-five per cent. of fire assay value at New York official quotation.

*Copper*.—All above 0.5 per cent. at electrolytic price per lb. less  $2\frac{7}{8}$  cents, provided lead under 3 per cent. If over 3 per cent., so that the ores are treated in lead furnace, the deduction is  $4\frac{1}{2}$  cents per lb. from electrolytic quotation per lb.

*Lead*.—All above 3 per cent. paid for at 90 per cent. of dry assay at the official price of desilverised lead, less 1 cent per lb. Under 3 per cent. not paid for.

\* *E. and M. J.*, Jan. 25, 1908, p. 223.

*Treatment Charge.*—Basal charge \$1 per ton, to which is added 10 cents per unit of insoluble in excess of iron (the same being credited for iron in excess of insoluble), 30 cents per unit for zinc over 10 per cent., 25 cents. per unit for sulphur in excess of 2 per cent. up to a maximum charge of \$1 per ton of ore, and 25 cents per unit of speiss produced in excess of 5 per cent.

At Mexican custom-smelting plants the basis of purchase varies somewhat from that in use in Colorado, as above. A few years ago the following schedule was in common use:—

<i>Pay</i> for silver, 95 per cent. of its value,	\$36.50 per kilo.
„ gold, $62\frac{1}{2}$ cents gold per grn., provided not less than 3.43 grms. per 1,000 kilos (= $1\frac{1}{16}$ oz. per ton).	
„ 90 per cent. of lead contents at $5\frac{1}{2}$ cents Mexican per kilo.	
„ excess Fe + Mn over $\text{SiO}_2$ per unit, . . . . .	15 cents.
<i>Charge</i> for excess $\text{SiO}_2$ over Fe + Mn, per unit, . . . . .	20 „
„ excess Zn over 8 per cent. per unit, . . . . .	55 „
„ excess S above 3 per cent., and up to a maximum of \$4.50 per ton per unit, . . . . .	3 „
„ excess As, Sb, or Ba above 3 per cent. added together, per unit, . . . . .	55 „

Federal taxes 3 per cent. on value of Au and Ag, 2 per cent. coinage tax on gold, all paid by purchaser.

The treatment charge was an elaborate sliding scale based on the percentage of lead alone, and ranging from \$20 per ton in the case of dry ores with less than 5 per cent. lead, down to 50 cents in the case of lead ores with 40 to 45 per cent. lead. Lead ores with over 50 per cent. of metal received a premium instead of a treatment charge.

The interesting article by Ingalls \* already referred to gives an approximation to the method by which the smelter arrives at the schedule upon which he purchases ores. The conditions assumed are those prevailing at the great Salt Lake and Colorado plants, the cost of smelting at which has been already given on a previous page as \$4.50 per ton of ore. It is now assumed that the refining and selling cost amounts to \$7.60 per ton of lead, without including amortisation or interest charges, and it is further assumed that on ore of the average grade treated the recoveries are as follows:—Gold 100 per cent., silver 97 per cent., lead 94 per cent., and copper 70 per cent. The grade of ore upon which the calculations are based is the average grade bought by the American Smelting and Refining Co. during the year 1902-3, which was as follows:—Gold 0.41 oz. per ton, silver 24.95 ozs. per ton, lead 9.87 per cent. (=197.3 lbs. per short ton), copper 0.959 per cent. (=19.17 lbs. per short ton). The value realised by the smelter on ore of this average composition was then—

Gold, 0.41 oz.	at \$20.56 . . . . .	= \$8.43
Silver, 24.95 ozs. $\times$ 0.97	at 0.5125 . . . . .	= 12.16
Lead, 197.3 lbs. $\times$ 0.94	at 4.11 . . . . .	= 7.69
Copper, 19.17 lbs. $\times$ 0.70	at 12.45 . . . . .	= 1.67
		<hr/> \$29.95 <hr/>

Taking the freight on products at a figure midway between the figures

\* *Min. Ind.*, vol. xvii., pp. 608-617.



actually paid at Salt Lake and at Denver, the total expenses incurred in handling this ore would be as follows :—

1. Smelting 1 ton of ore	at \$4.50	. =	\$4.50
2. Converting 40 lbs. copper matte	at 0.7	. =	0.28
3. Freight on 190 lbs. lead bullion	at 0.43	. =	0.82
4. Freight on 13½ lbs. copper in products	at 0.5	. =	0.07
5. Refining 190 lbs. lead bullion	at 0.38	. =	0.72
6. Refining 13½ lbs. copper bullion	at 0.7	. =	0.09
7. General expense reckoned at per ton of ore,	.	. =	0.40
8. Amortisation	" "	. =	0.25
9. Tie-up of metals ( <i>i.e.</i> , interest),	" "	. =	0.30
10. "Metal account" ( <i>i.e.</i> , insurance against depreciation),		=	0.30
<hr/>			
Total cost to smelter per ton of ore,	.	. =	\$7.73
Allow for smelters' profit,	.	. =	2.00
<hr/>			
Total,	.	. =	<u>\$9.73</u>

The gross value, \$29.95, less the total of \$9.73, leaves \$20.22 for the value of the ore and freight upon it to smelting works.

The trial schedule made to ascertain the treatment charge properly made on a neutral ore of the composition mentioned, under the conditions supposed, would be as follows :—

Gold at \$19.50 U.S. Cy. per oz., silver 95 per cent. of the New York quotation, lead 40 c. per unit, copper N.Y. quotation less 7 c. per lb.; which works out as follows :—

Gold,	0.41 oz.	at \$19.50,	. . . .	\$8.00
Silver,	24.95 ozs. × 0.95	at 0.50½,	. . . .	11.93
Lead,	197.3 lbs.	at 0.2,	. . . .	3.95
Copper,	19.17 lbs.	at 5.45,	. . . .	1.04
				<hr/>
				\$24.90
Deducting the sum already calculated as that available for purchasing the ore,				20.22
				<hr/>
Available for treatment charge,				<u>\$4.68</u>

The schedule quoted to producers would, therefore, be as above indicated, with the addition of the above words, "Treatment charge \$4.68 per ton, on neutral basis, d/d at works."

## CHAPTER XIII.

### FLUE-DUST—ITS COMPOSITION, COLLECTION, AND TREATMENT.\*

#### 1. COMPOSITION.

THE term "flue-dust" is commonly applied to a product which, collected very often in a single series of apparatus, is in reality composed of a mixture in varying proportions of two distinct classes of material. The first is merely fine dust of ore and of fuel carried over mechanically by the draught; the other, more properly called lead-fume, being a mixture of sulphide, sulphate, and oxides of lead with some zinc oxide and sulphite, arsenious acid, and other substances, volatilised in the hottest part of the furnace, and subsequently solidified on cooling into an infinitesimally fine dust. The individual particles are so fine that it would be almost impossible to collect them but for their mutual attraction, which causes them to form flaky or feathery aggregates, somewhat as snow crystals combine to form snowflakes. The flue-dust collected nearest the furnace contains most of the particles of ore mechanically carried over, while that farther away approximates to the composition of a true fume.

In the product collected in the flues serving ore hearths and reverberatories, fume largely predominates, partly because the volatilisation loss in these types of furnaces is so much greater, partly because, the ores treated being purer, such fine material as is carried over by the current of furnace gases is almost pure lead sulphide, oxide, and sulphate. In the blast furnace, however, with its poorer ores and large amounts of flux, &c., the material collected, at any rate in the first portion of the flue, consists largely of fine carbonaceous and siliceous material, lime, iron ore, &c.

The flue-dust from blast furnaces run with a cold top is always dark in colour, owing to the presence of particles of fuel in the dust-chambers near the furnace, and to the presence of large quantities of volatilised PbS at a greater distance along the flue. The marked difference in composition between mere chamber dust and true fume is well exemplified by the analyses of the products of the Lone Elm hearth furnace given in Chapter iv., p. 63, and the difference is quite as marked in the case of blast furnaces. The flue-dust from roasting furnaces contains little or no ore and fuel, approximating more to the character of a fume, while the fume from furnaces in which the ore is fused or slagged is naturally much richer both in lead and silver than when the ore is plain roasted.

The composition of various samples of flue-dust is given in Table XXXVIII. None of the analyses show the presence of ammonia, which, however (derived

\* Consult Hering, *Die Verdichtung des Hüttenrauches*, Stuttgart, 1888; also Hofman, *Metallurgy of Lead*, 1896, pp. 378, *et seq.*; and Iles, *S.M.Q.*, vol. xvii., pp. 97, *et seq.*

Reference,	1	2	3	4	5	6	7	8	9	10	11	12	Globe, Denver, Colo.		15
													Furnaces.	Sing Settlers.	
Pb.	46.54	62.26	44.80	27.90	34.8	20.26	67.04	60.15	..	23.65	39.45	37.65	25.60	..	16.2
PbO.	..	..	..	..	18.0	..	..	..	..	57.31	..	..	..	21.64	..
PbSO <sub>4</sub>	..	..	..	..	2.9	..	..	..	..	..	..	..	..	14.64	..
PbS.	4.87	1.05	..	..	1.0	..	..	..	71.13	..	3.70	..	4.74	..	..
Zn.	..	..	..	..	1.5	28.90	4.22	10.50	4.17	..	..	5.32	0.95	21.00	3.0
ZnO.	1.60	1.60	4.80	..	..	0.72	tr.	..	..	..	..	tr.	..	..	..
CuO.	..	..	1.52	..	..	..	..	..	..	..	..	..	..	..	..
Bi <sub>2</sub> O <sub>3</sub>	..	..	..	..	..	..	0.31	4.30	1.89	4.40	..	..	..	..	..
SnO <sub>2</sub>	..	..	..	..	..	..	0.16	3.25	..	0.54	36.80	..	..	0.18	..
As <sub>2</sub> O <sub>3</sub>	..	..	..	..	4.0	..	..	..	..	..	tr.	..	0.93	..	..
As <sub>2</sub> O <sub>5</sub>	..	..	..	..	..	..	..	..	..	..	..	..	13.40	..	22.10
Mn.	..	..	..	..	1.4	..	..	1.30	8.54	11.84	..	..	..	1.00	..
Fe.	..	..	..	..	..	..	1.00	..	..	..	tr.	24.98	..	1.20	..
Fe <sub>2</sub> O <sub>3</sub>	4.16	3.00	tr.	..	4.5	..	..	..	3.64	0.92	tr.	1.31	..	0.60	6.00
Al <sub>2</sub> O <sub>3</sub>	..	..	10.00	..	..	0.45	0.61	..	0.34	0.17	tr.	5.26	5.30	..	..
CaO.	6.07	3.77	..	..	..	..	..	..	..	..	..	..	..	..	..
MgO.	..	..	..	..	..	..	..	..	..	..	..	..	4.60	..	..
BaO.	..	..	..	..	..	18.14	..	..	..	..	0.16	8.63	26.20	1.70	19.0
SiO <sub>2</sub>	10.12	1.97	9.00	..	12.3	5.43	5.42	..	6.70	..	5.73	2.53	4.20	..	2.4
S.	..	..	..	..	7.8	..	14.07	18.60	..	0.90	1.75	1.61	..	0.75	..
SO <sub>2</sub>	26.51	25.78	28.81	13.00	2.8	..	..	..	..	..	..	..	..	0.60	..
SO <sub>3</sub>	..	..	..	..	..	..	..	..	..	..	..	11.20	..	..	..
CO <sub>2</sub>	..	..	..	..	..	9.58	..	..	..	..	..	..	..	..	..
H <sub>2</sub> O.	..	..	..	..	2.5	..	5.80	1.60	3.17	..	..	0.04	..	24.40	0.098
C.	..	..	..	..	..	0.07	0.003	..	..	..	..	..	..	..	..
Ag.	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Ag ozs. per ton,	99.87	99.43	101.96	99.53	93.5	..	98.633	99.70	99.58	99.73	..	98.53	..	..	..
..	..	..	..	..	..	..	..	..	..	..	1 oz.	..	33.4	..	28.6
..	..	..	..	..	..	..	..	..	..	..	..	..	0.14	..	0.108

References.—1 and 2. Percy, *Metallurgy of Lead*, p. 451. 3. French, *Eng. and Min. Journ.*, Jan. 17, 1880. 4. Hering, *Die Verdringung des Hüttenrauchs*, 1888. 5. Belling, *Metallhüttenkunde*, p. 87. 6. Schnabel, *Handbuch der Metallhüttenkunde*, p. 389. 7. Eggleston, *Trans. A.I.M.E.*, vol. xi., p. 397. 8. Percy, *Metallurgy of Lead*, p. 453. 9 and 10. Sanchez Massia, *Metallurgia del Plomo*, Madrid, 1893, p. 293. 11. *Private notes*, 1904. 12. Dewey, *Bull. 42 U.S. Nat. Mus.*, p. 53. 13 and 14. Iles, *Eng. and Min. Journ.*, Dec. 2, 1899, p. 669; and *Metallurgy of Lead*, 15. Hahn, *Trans. I.M.M.*, vol. vii., p. 274.

doubtless from the coke), is frequently present in blast-furnace flue-dust in the form of sulphate, as pointed out by Iles.

It will be noticed that a considerable proportion of PbS exists in most of the samples. The volatility of PbS at moderate temperatures, much below its fusing point, has been already referred to in Chapter i., and hence it is to be supposed that all the lead existing as PbS in flue-dust was volatilised as such, while that existing as PbO was probably volatilised as metal in the lower portion of the furnace. When, as frequently happens, the deposited fume catches fire and burns in the flue on account of some furnace being barred down or blown out, the residual matter consisting of sulphates and oxides, acquires a white or grey colour. This is always the case with fume from reverberatory furnaces. When the fume has not ignited spontaneously it is frequently set on fire before removal from the flues, in order to convert it into a crisp coherent crust or cinder, which is less liable to dusting, and forms a better material for subsequent treatment than if it remained in the pulverulent condition. Fume collected by filtering methods is, as will be seen, always ignited prior to withdrawal from the bag-houses. The lead in such ignited fume, and in fume from ore hearths, reverberatories, and roasters, from all of which the gases leave at a high temperature, is found chiefly in the form of a basic sulphate.

Careful testing of the flue-dust from different parts of a condensing system brings out the fact that the ratio of silver to lead generally, although not invariably, diminishes with increased distance from the furnace. This is only what might be expected from the less volatility of silver compounds compared with those of lead. Thus, at Bms the silver in the flue-dust and fume diminishes gradually from nearly 5 ozs. down to 1 oz. per ton. At Murray that portion of the flue-dust collected in the first dust-chamber near the furnaces contains 22 ozs. Ag per ton, whereas the portion collected nearest the stack contains only 8 ozs. per ton. The true fume filtered out from blast-furnace gases, after all the suspended fine particles of ore have been settled out, is always poor in silver, containing not over 2 to 4 ozs., even when the flue-dust from near the furnaces contains 30 ozs. silver or over.\* Flue-dust is always relatively poorer in silver than the ore from which it was derived, for the same reason; and fume-lead (obtained as its name implies chiefly from the true fume or finer portion of flue-dust) is harder and more impure than ore lead as well as poorer in silver, because such substances as arsenic and antimony present in the ore furnace are mostly sent into the fume.

## 2. FLUE-DUST COLLECTION OR CONDENSATION.

The two varieties of flue-dust—viz., true dust and lead-fume—differ greatly as regards the difficulty of collecting them. The former only requires sufficient reduction in the velocity of the air current in order to settle out almost completely, even without thorough cooling, and this reduction in velocity is best secured by enlarging the diameter of the flue, and especially by constructing large dust- or settling-chambers.

\* According to a private communication from L. Pitblado, when smelting rich telluride gold ores, the proportion of gold in the fume increases with the distance from the furnace—namely, from 0.24 oz. up to 0.324 oz. per ton. This may be due to the tellurium volatilised carrying with it a volatile telluride of gold.

The readiness with which the particles of dust are settled out by reducing the velocity of the current of gases is shown by some experiments of Shelby \* on sizing average samples of flue-dust from the *Cananea* plant. At this plant the flue-gases from eight blast furnaces pass first for an average length of 100 feet through sheet-iron flues varying from 100 to 150 square feet in sectional area, and then enter a dust-chamber 180 feet long, the cross-sectional area of which is 1,000 square feet. The proportion of coarse to fine particles in the material collected at various points is shown below :—

Sectional Area of Flue.	Distance.	On 100 Mesh.	On 200 Mesh.	Through 200.
Square feet.	Feet.	Per cent.	Per cent.	Per cent.
100 to 150	50 to 150	86·4 to 79·1	11·9 to 18·5	1·7 to 2·4
1000	0 to 90	77·6 to 50·7	18·5 to 40·7	3·9 to 8·6
1000	90 to 180	23·9 to 5·3	56·9 to 56·6	19·2 to 38·1

So that, after passing through only 180 feet in length of the large dust-chamber, the proportion of particles of over 100 mesh in the product is reduced from nearly 80 to about 5 per cent.

The total weight of coarse particles collected in ordinary blast-furnace work is so much greater than that of the fine particles, that from one-half to two-thirds of the total flue-dust saved is caught in the first portion of the flue, say, the first 100 to 200 feet. This first section of flue, therefore, in all modern plants is always made hopper-shaped—that is, whether built of brickwork, of re-inforced concrete, or of sheet iron, the bottom is formed into a number of little hoppers with closely fitting doors, and the whole flue is raised from the ground high enough to permit of the passage of suitable cars below the hoppers.

The floating particles of true fume are so minute that, even after a thorough cooling (without which no condensation is possible), mere settling has but little effect upon them. A better means of condensation is to increase the length of the flue, and alter frequently the direction of the motion by means of baffle walls, so as to increase the number of contacts with the sides of the flue, to which a certain proportion of the floating particles adhere, and from which they fall to the floor.

It is found, however, that both of these means are comparatively inefficient for condensing true lead-fume, and accordingly at all the older lead works the length of the flues was continually increased, with the result that each successive addition yielded sufficient lead to more than pay interest on the capital expended in its construction. In this way, for example, the flues at Freiberg have grown till they now attain the enormous length of 5 miles, the sectional area being for the most part 40 square feet ; again, at Mazarron, Spain, and at Laurium, Greece, the flues are 2 miles long, and are being continually added to.

Where space is scarce, or the ground unsuitable for building long single flues, “ zig-zag ” flues in blocks are often constructed like those of the *Cordoba* Smelting Works shown in Fig. 201.† The advantage of this construction (apart from convenience where space is limited) is that it saves nearly one-half

\* *E. and M. J.*, Jan. 25, 1908, p. 205.

† *Private Notes*, 1908.

of the walling otherwise necessary, since each longitudinal wall, except those on the extreme outside of the block, serves for two flues; on the other hand, the flues are rendered much more awkward to get at, and cannot be cleaned without a shut-down of the entire plant. The flues shown in Fig. 201 are 4 feet 3 inches wide by 5 feet 9 inches high, but the same principle may be adopted for flues of much larger sectional area.

The methods adopted for the condensation of the finer portion of flue-dust and of true lead-fume may be considered as falling under the two separate stages of *cooling* and *collection*.

**Cooling.**—The first essential in condensing lead-fume is that the gases shall be thoroughly cooled, not, however, on account of any trifling amount of lead which may possibly remain in the condition of vapour, for the vapour tension of lead or its compounds, even at the boiling point of water, is quite imperceptible.

According to the experiments of Iles,\* the average temperature of the gases escaping from a modern lead blast furnace in good work is about 214° F., the maximum registered being 335°, when the accretions had reached a

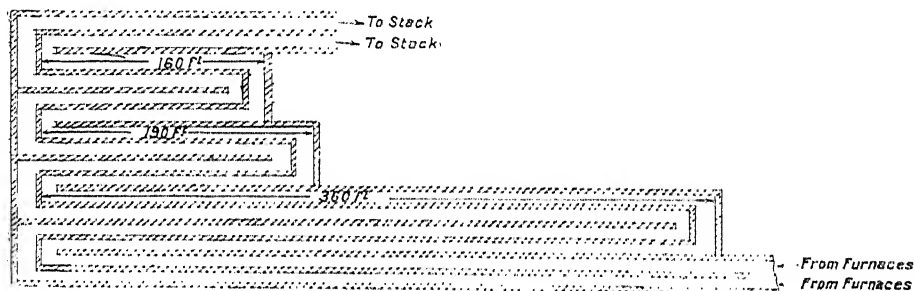


Fig. 201.—Zig-zag Flues (Plan).

considerable thickness. When the heat is allowed to creep up, however, and the furnace has a "hot top," the temperature may be much higher.

Thorough cooling is necessary, chiefly because minute hot particles repel instead of attracting each other, and, therefore, until the gases are brought down to a temperature below that of boiling water, no collection of the particles into flaky masses can take place. The temperature of furnace gases from lead blast furnaces is, according to the above figures, not usually high, and cooling, therefore, is readily effected by passage through a comparatively short length of sheet-iron flue simply air-cooled on the outside; but with gases from roasting furnaces and from reverberatories, which on entering the flue are still almost red hot, special means of cooling are essential.

**Air Cooling.**—A common method of air cooling is by means of a sheet-iron flue, as shown in Figs. 202 and 203. The former represents the flue in use at *Richmond* (Nevada), which was 800 feet long, and shows the mode of suspension to a wooden trestle; the sliding doors, D, were 2 feet apart.

A better arrangement is that shown in Fig. 203, where the flue-dust collects in the hopper-shaped portions which end in short square pipes closed

\* *S.M.Q.*, vol. xvii., p. 19.

by sliding doors, and which can be conveniently emptied into a closed truck by means of a movable length of pipe. This flue can be suspended by means of short trestles underneath the upper body, leaving the hopper parts hanging free. An arrangement of hoppers which is easier to make is that in use at *Ems*, but the hoppers are wedge formed, instead of pyramidal, and are not nearly so easy to clean. All sheet-iron flues should be kept well painted with some form of asbestos or "silico-graphite" paint.

Instead of a rectangular section, economy of material is nowadays secured by the use of what is called a "balloon" flue—namely, a sheet-iron flue of circular or oval section with suspended hoppers at short intervals, similar to those depending from the square flue of Fig. 203. These balloon flues serve the double function of cooling the gases and of settling out the bulk of the coarse dust particles, and they are now commonly made of large size; for instance, 15 feet in diameter if circular, or 12 feet horizontally by 16 feet vertically if oval in section.

Another air-cooling arrangement is that in use at the *Grant Works*, Denver, where, owing to want of space, underground flues and chambers have been built of hollow terra-cotta tiles, through which a current of air is made to circulate by means of a separate chimney.

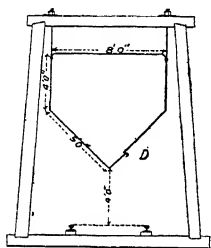


Fig. 202.—Cooling Flue.

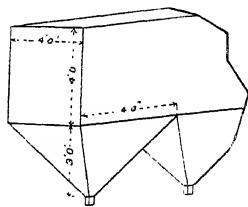


Fig. 203.—Cooling Flue.

**Water Cooling.**—Simple air cooling requires very long flues, and, therefore, many attempts have been made to utilise water for accelerating the rate of cooling. When applied internally, water also assists in condensing the dust, but this plan has many disadvantages, as will be seen under the head of *Wet condensation*. The external use of water for condensation is exemplified by the *Hagen* appliance in use at Freiberg. This is made of sheet lead; it is suspended like an ordinary sulphuric acid chamber, and cooled by a succession of shallow leaden tanks attached respectively to the roof and floor, and connected by elliptical sheet lead pipes burnt on to the sides of the flue and to each other. Water circulates alternately up and down through this series of tanks and pipes; the appliance is said to be highly efficient in cooling the gases, though too costly for ordinary use.

**Collection.**—The collection of flue-dust, of course, begins before the gases have been properly cooled, and further cooling takes place during the collection. This further cooling is important, because the dust-laden gases are repelled by a surface which is warmer, and attracted by one which is at a lower temperature than their own. The preliminary partial cooling is, however, essential.

After cooling, the most important requisites for condensation are:—(1) To reduce the velocity of the air current; and (2) to expose as large a condensation surface as possible to the gases. Simple increase in the length of the flue fulfils to some extent the second condition, but not the first; large settling chambers fulfil the first condition, but not the second. Of the two requisites, the second is of much greater importance than the first. It is found that in a horizontally partitioned flue very much more flue-dust is deposited in the upper than in the lower half in a given time; also that the point of maximum velocity is on the centre line of the flue at about seven-tenths of the total height from the bottom. At the bottom the forward velocity is very slight, or there may even be a slight eddying or return current, yet not so much is deposited here as on the top, where the velocity is much greater. In an ordinary flue without horizontal partitions the deposit on the bottom constantly accumulates, because the flue-dust caught on the top and sides falls from time to time in masses of greater or less size and remains there.

Increase of surface, however, without previous cooling has but little effect. Aitken has shown \* that a hot surface *repels* floating particles, especially if moist; whereas a surface which is slightly cooler than the floating particles *attracts* them. Increasing the surface acts also in another way—viz., by increasing the friction of the current of gases, by which their velocity is considerably reduced within an inch or two of the surface, and the tendency of the particles to settle out can better come into play.

**Construction of Flues and Settling Chambers.** †—Both flues and settling, or dust, chambers are commonly built of brick, tied together by means of iron rods to avoid leakage. The parts near the furnaces are always built of firebrick, or of sheet iron as already described; but, after thorough cooling of the gases and condensation of most of their contained moisture, ordinary brick or stone may be used for the remaining portions. Substantial construction is essential if the flues are to stand for any time, and thus a properly designed system becomes very expensive. Two very good forms of flue are shown in Figs. 204 and 205. The former ‡ represents the flue of the Montana Smelting Company at one of the manholes 50 feet apart; the cleaning doors and pits shown in this figure occur on alternate sides at intervals of 9 feet 4 inches, and the method of bracing with wooden posts set into a masonry foot and tie-rods through the springing of the arch is plainly shown. Fig. 205 is a section of the new flues at the *Em's* Smelting Works. Arched flues were formerly used here, like those in Fig. 204, but they were found to crack a good deal, and flues of the form shown in the figure in which a course of iron rails across the side walls supports brickwork have, to a great extent, replaced them.

A similar form of flue, which for small sizes has proved in the experience of the author and others very stable in practice, owing to the freedom from roof-thrust upon the walls, is one in which the roof is formed of flat arches of about 4 feet span and only half-a-brick thick, rising from strong  $\perp$ -iron or small I-beams extending transversely across the top of the walls. For flues 4 feet wide a 4-inch  $\perp$  suffices, and for 5 feet span a 5-inch I will serve. Sup-

\* *Proc. Roy. Soc. Edin.*, vol. xxxii., p. 239.

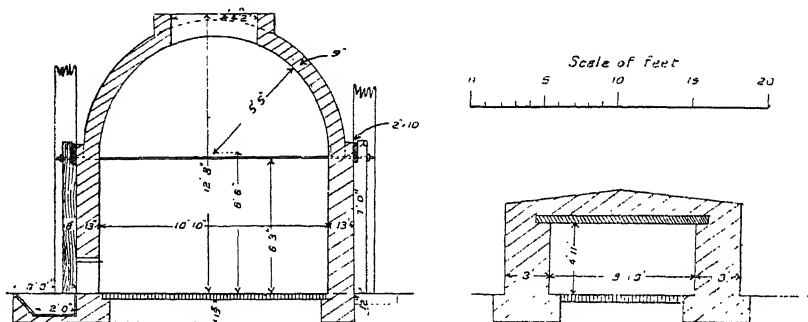
† Sometimes called "Sedimentation chambers."

‡ Borrowed from Hofman, *op. cit.*, p. 289.



port to the walls is given by outside buttresses at intervals. Such flues, if well plastered on the inside with plaster of Paris, or with a mixture of tar and sand boiled together and well-worked with the trowel have, according to the author's experience, a fairly long life under even the most trying conditions—namely, the handling of wet acid fumes after a partial condensation by water. Tar plastering, however, cannot be used in flues which are exposed to temperatures much in excess of that of boiling water.

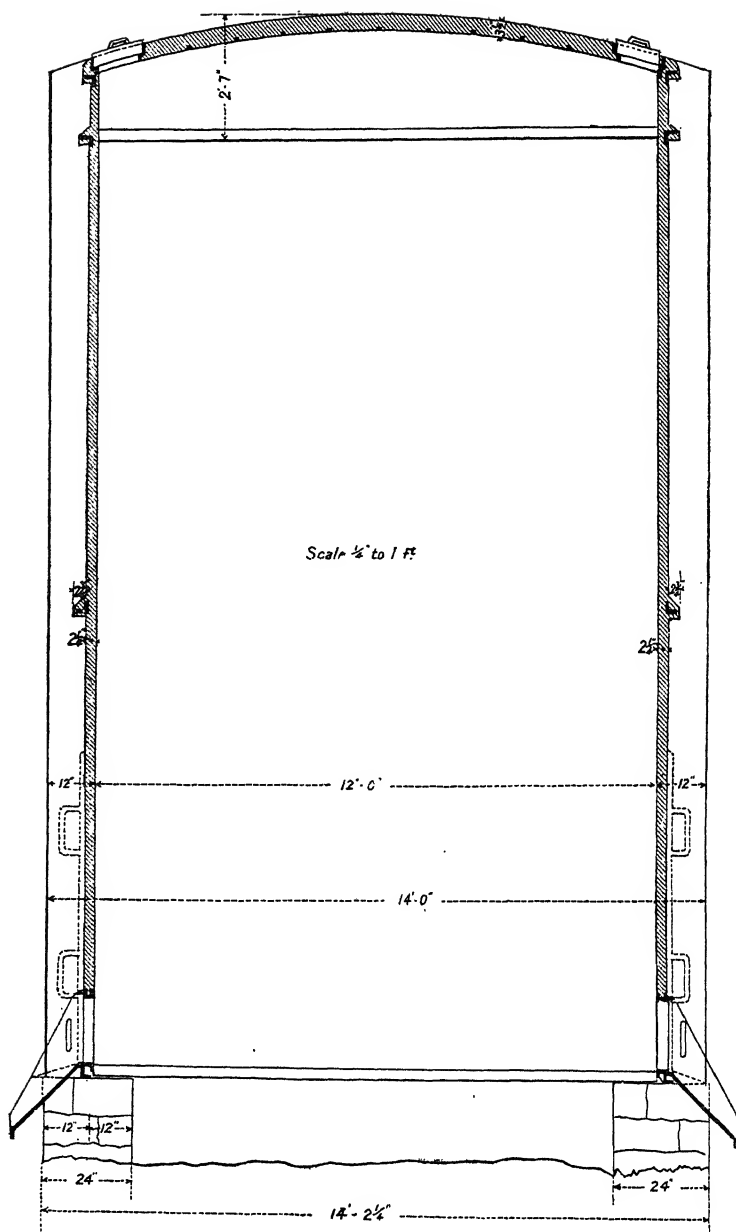
Baffle walls in the flues, forcing the gases to take a tortuous course and largely extending the surface, are found to greatly increase the amount of flue-dust collected, though somewhat diminishing the draught. It is best to run up these baffle walls from alternate sides so as to cover about one-half of the sectional area of a large flue, and to supplement them, as at *Aurora* and other places, by 2-feet walls built up from the floor at intervals, so as to form pockets, which prevent the dust from drifting along the flue or being again caught up by the current. Where brick cannot be obtained, stone may be used for building flues, but they are not then so strong, and much more mortar is required. Moreover, the more open mortar joints allow moisture to penetrate readily, and the walls are more liable to be destroyed by the effect of dilute sulphuric acid on the mortar.



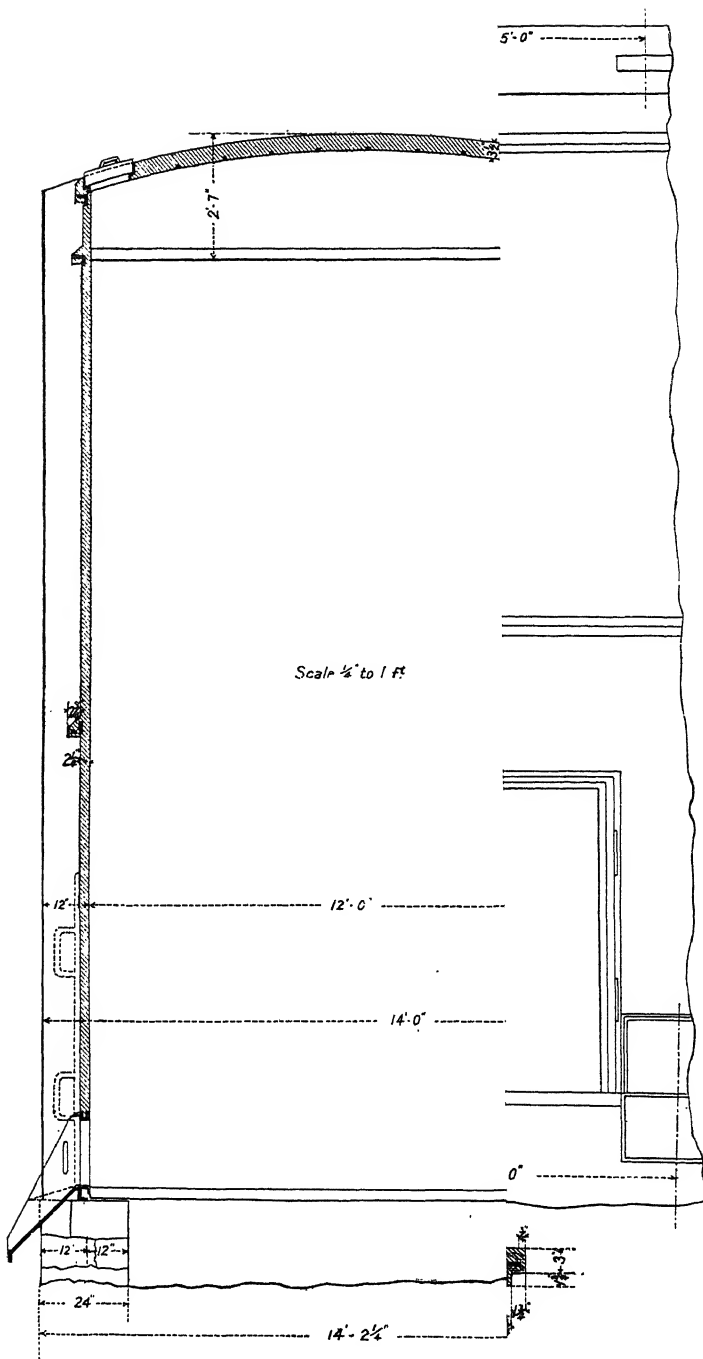
Figs. 204 and 205.—Brick and Masonry Flues.

Occasionally, in countries where water for mortar mixing is scarce, flues have been excavated in the solid rock, and lined with a plaster of cement mortar, or with a lining of brick set edgeways. This method of construction, however, will rarely be much cheaper than that of ordinary flues above ground, and it possesses the great disadvantages of not being readily accessible for cleaning purposes without shutting down the furnaces, and of not favouring any cooling of the gases. In old-established works, however, flues have frequently to be built entirely underground on account of want of space.

Of late years re-inforced concrete has been somewhat largely employed for the construction of flues and dust chambers, for which purpose its lightness and the thinness of the walls render it specially suitable. A flue on this principle (an open network of coarse iron wire imbedded in cement concrete) was erected in 1890 by the Victor Friedrichshütte (Tarnowitz, Up. Silesia), 1,640 feet long, 6 feet 7 inches wide, and 9 feet 10 inches high, and is said to have given satisfactory results. A group of condensing chambers on this system was built in 1893 at Freiberg, at an addition to the flues from the



Figs. 206, 207, and 207A.—



arsenic and zinc works. It consists of six parallel flues, each 100 feet long, 10 feet 4 inches wide, and 10 feet 10 inches high, giving a total extra length of 686 feet of flue, with 2,086 cubic metres capacity.

The lattice work is composed of vertical and horizontal wires,  $\frac{9}{32}$  inch apart, the wires being  $\frac{9}{32}$  inch and  $\frac{3}{16}$  inch diameter respectively. The concrete is composed of one part cement, three parts quartz in fragments of about  $\frac{1}{2}$  inch diameter, and two parts quartz tailings from the dressing floors. The walls are built 2 inches thick, with the network in the middle; they are painted on the outside with coal tar and on the inside with a silicate paint. The cost was £1,330, as against £1,500 which it would have cost in light brickwork, and £2,500 in sheet lead properly supported.

Large concrete flues, and especially dust-chambers, when built of re-inforced concrete, should have at intervals, in order to secure the necessary stiffness, either substantial concrete pillars containing iron rails or angles to which the metallic network is attached; or else, complete skeleton framework of light angles, to which are rivetted plain bars carrying the expanded metal laths upon which the concrete is simply plastered, the mixture containing cement in the proportions of about 1 : 3 for the inside coat, to perhaps 1 : 5 in that outside, taking care in either case that the aggregate is not too coarse. The first mode of construction is exemplified in a paper by Edwards,\* who gives 4 inches as the minimum thickness of concrete which it is advisable to employ. The second method of construction is exemplified in Figs. 206 to 210, which give details of the construction of a large dust chamber at *Mapimi* (Mex.).

The chief troubles with re-inforced concrete as a material for flues are two—namely, the tendency to crack from expansion and contraction caused by the difference between the night and day temperature acting upon such thin layers of non-homogeneous material, and the corrosion caused by acid fumes. The tendency to crack from expansion and contraction is, of course, much aggravated by excessive thinness of the walls, and, on the whole, it is probable that the majority of the failures reported have been from this cause. Flues on the so-called Monier principle, for instance, built at the *Philadelphia* Works about 1896 were not over 2 inches thick, and trouble seems to have been experienced in every instance where flue- or dust-chamber walls have been built with less than 3 inches of concrete as an absolute minimum, so that it would seem to be preferable to adopt Edwards' limit of 4 inches as a safe thickness, at any rate in very large structures. The formation of transverse cracks caused by longitudinal expansion and contraction in the case of long flues may be to some extent combated by building the flue in short sections with a  $\frac{1}{2}$ -inch space between each section, which is covered by a strip of thin flat iron forming a rough expansion joint. The second difficulty is more serious, for although, in the portion of the flue near the furnace, where the fumes are still above the boiling point of water, little harm would be expected, it is found that so soon as the temperature of the inside of the flue falls low enough to permit the condensation of moisture, corrosion sets in, the moisture in the gases combined with that penetrating from the outside, forming sulphurous and sulphuric acids. These acids once formed rapidly decompose the cement, while the crystallisation of calcium sulphate, zinc sulphate, and other soluble salts in the pores still further disintegrates the

\* *Trans. Amer. Inst. M.E.*, vol. xxxv., pp. 60-81.



the cement attacking the oil. A mixture of asbestos and water-glass is said to have given satisfaction in some instances, and so has ebonite varnish; well-boiled coal-tar applied hot with a mop will, according to the author's experience, render a good cement plaster impervious to acid fumes for a long time, provided the application is thoroughly rubbed into the pores, and two or three coats are given both outside and inside the flue.

On the whole, however, the behaviour of re-inforced concrete as material for the construction of flues and dust-chambers does not appear to have been altogether satisfactory, and of late there has been a reaction in favour of ordinary brick, or of rubble masonry with brick lining well laid in clay mortar, lime, of course, being only employed for the outside work. A good construction, then, for ordinary dust-chambers or large flues is, first, a foundation of concrete 1 : 4 : 6, then 9-inch brick walls laid in clay mortar inside and good lime mortar for about  $1\frac{1}{2}$  inches outside, with pilasters at intervals for additional support, strips of hoop iron or expanded metal being laid between the courses of brick at intervals. Roof of iron girders covered with corrugated iron, having a thin roof of good cement concrete rammed on top, or half-brick flat arches running transversely to the flue with steel girder abutments as already described. All ironwork inside flue liberally painted with three coats of hot boiled coal-tar; all plaster, brickwork, and roof outside to have two coats of the same material applied hot with tar-mops to prevent moisture from leaking through.

At the *Murray Works* (Utah) concrete flues at first built have been replaced by flues of 250 square feet section, built of brick only 9 inches thick, in the form of an inverted catenary arch, which requires neither abutments nor binders. In order to counteract the tendency to destruction caused by longitudinal expansion and contraction, expansion joints, 3 inches wide, are left at every 20 feet in length.

**Methods of Increasing Surface.**—In addition to baffle walls, the principal methods of increasing surface are the following :—

1. **Cowper Flues.**—The principle upon which these work is easily understood from a reference to Fig. 211, which shows a plan of one set of such flues. The main flue from the furnaces has a stopping door, D, fixed in it,

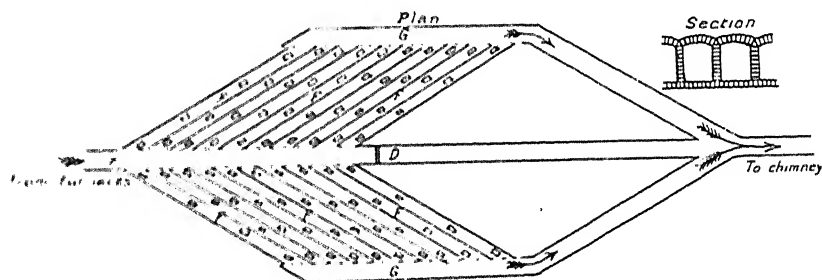


Fig. 211. Cowper Flues.

and a number of parallel diagonal flues branch off from it on either side, the opening into each from the main flue being very small, so as to divide the current of gases into approximately equal portions, which are again collected into the main flue by the gathering flues, G, on either side. In this way the

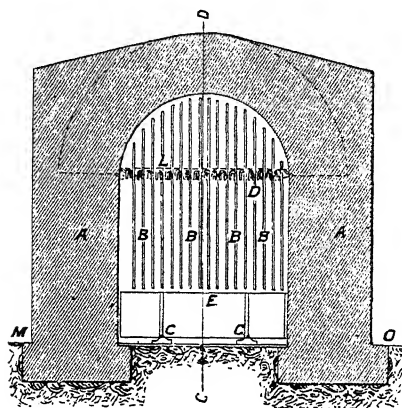
velocity of the current in the parallel flues, which are narrower than the principal flue, is reduced to, say, one-tenth of that in the latter; while the surface exposed for condensation is twenty times as great.

An installation of this kind has been in use at *La Tortilla Works, Linares* (Spain) since 1882, working most satisfactorily, and yielding in six months over 90 tons of fume, containing 67 per cent. lead. The Cowper flues possess

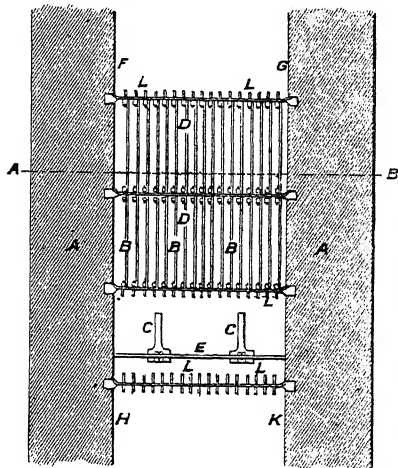
the very great advantage that the velocity of the current is reduced *simultaneously* with the increased surface offered to the gases, and, therefore, the fumes are under the most favourable conditions for deposition.

**2. Brushwood and Coke Towers.**—These methods were formerly largely employed, usually with the addition of a spray of water, but the product obtained is always in a very unsatisfactory form for handling, not to mention the great obstruction to the draught; they are now practically out of use.

**3. Freudenberg Plates.**—The Cowper flues may be multiplied and the individual flues narrowed to any desired extent, but a cheaper and simpler means of attaining the same end as regards increase of surface is by the adoption of Freudenberg plates shown in Figs. 212 and 213.\* At the level of the springing of the arch flat iron bars, L, pierced with holes about 4 inches apart, are fixed across the flue by twisting the ends at right angles so as to enter the brickwork, and iron pins are driven into the holes so as to project about 1 inch on each side. Sheets of iron, B, No. 22 gauge, have pieces, D, rivetted to the ends of their upper side, in which loops are formed which drop on to the pins and so support the sheets in a vertical position 4 inches



Scale 1:50



Figs. 212 and 213.—Freudenberg Plates in Flue.

apart. Old sheet iron can be used for this purpose; in fact, old jig screens are commonly employed, while at the extreme end of the flue system where the gases are cool, pasteboard has been found to answer when the flue-dust is not ignited before removal. The flue-dust collecting on the plates falls to the ground on attaining a certain thickness, and at every 15 to 18 feet a cross.

\* Egleston, *Trans. A.I.M.E.*, vol. xi, p. 379.

partition, E, is erected in the lower part of the flue to prevent dust once collected from being taken up again by the air current and carried away into the stack. These cross partitions are of sheet iron bolted to cast-iron feet, C, and reach up in the flue about 20 inches.

The plates may be placed much closer together, but they are then liable to choke up, and the friction is so much increased that unless the chimney is of great height and the flue runs uphill to it recourse must be had to increasing the chimney draught by artificial means. When 4 inches apart, the ordinary draught is but slightly interfered with.

A good arrangement is to provide two or three parallel lengths of flue fitted with the Freudenberg plates, so that the current of gases can be either split up among the various flues, or shut off at pleasure from one or the other for the purpose of cleaning, which has to be done so often that, in the absence of some arrangement of the kind, the furnaces may have to stop to allow of cleaning out the flues.

It might be supposed that thin iron sheets would rust very quickly, but this is not found to be the case where the gases are properly cooled, and where they pass through a long length of flue before reaching that part where the plates are placed, so that all the moisture is condensed out of them before they come in contact with the iron.

An installation of Freudenberg plates put in at the Muldner Hütte (Freiberg) in 1890 is described by Bauer \* as a series of twenty-one parallel flues, each 33 feet long, forming part of the condensing system of the blast furnace and refining departments. The total surface of these flues is 1,520 square metres; they are constructed of iron sheets fixed about 20 cm. apart. The sheets above the springing of the arch are fixtures, those in the central portion of the flue being suspended like those at Ems. The result of the insertion of the sheets was that, with an increase in ore smelted of 15 per cent., the flue-dust collected in the whole system showed an increase of 45 per cent.; the increase in that portion of the main condensing chamber containing the flues was no less than 82 per cent., while nearer the stack the quantity condensed was actually less than before. Unfortunately, there is no excess of chimney draught as at Ems; in fact, a Guibal fan has always been required to assist the natural draught, and it was found that an increase of 40 per cent. in the work of the fan was required to overcome the increased resistance caused by the plates. The Guibal fan used is 20 feet in diameter and 3 feet 6 inches wide; it takes about 7 H.P. to drive it at the rate of 65 to 85 revolutions per minute, which gives a pressure equal to 34 mm. ( $1\frac{3}{8}$  inches) of water, about 20 mm. of which corresponds to the resistance introduced by the plates. The cost of 20 mm. pressure is about £250 per annum, which, together with the interest and depreciation on the plant, must be set against the saving effected.

Kroupa † describes the installation in a dust-chamber of hanging plates of corrugated galvanised iron, which expose a very large surface to the gases. The dust falls off the plates at intervals into hoppers in the bottom of the chamber, which deliver into a pair of screw conveyors, so that cleaning the flue becomes an automatic process.

4. **Rösing Wires.**—Instead of sheets of iron, wires may be used as first

\* *Jahrbuch für das Berg- und Hütten-wesen im K. Sachsen*, 1894, p. 39.

† *Oesterreichische Zeitschrift*, 1905, liii., 347, quoted in *Min. Ind.*, vol. xiv., p. 417.



tried by Rösing.\* The wires used are of No. 10 gauge, and three-fourths the full height of the flue; they hang in parallel rows from a wire netting of 1 inch mesh, which is rivetted to I-beams running across the top of the flue, as shown in Fig. 214. At the *Tarnowitz Works* an installation on this system in 1888 reduced the total amount of solid matter escaping during the year from 54 to 40 tons per furnace, without perceptibly influencing the draught, the extra material collected being very rich in lead.

At *Great Falls* the new dust-chamber is 478 feet long by 176 feet wide and 21 feet high, divided lengthwise into two halves, either of which can be shut off at will. It is built entirely of steel columns and I cross-beams, with brick panels; the gases being dry have no effect upon the brickwork, which is left exposed without protection. The floor of the dust-chamber is formed into hoppers, and, in the basement below these, trains of electrically-operated dump-trucks remove the flue-dust from time to time. Fastened to the I-beams forming the roof is a heavy netting of copper wire, from which iron wires are hung 4 feet apart, in order to increase the surface available for dust deposition. The wires are all connected in sections below their point of support, and the connecting framework is shaken by means of a beam extending outside the house, operated by a small motor.

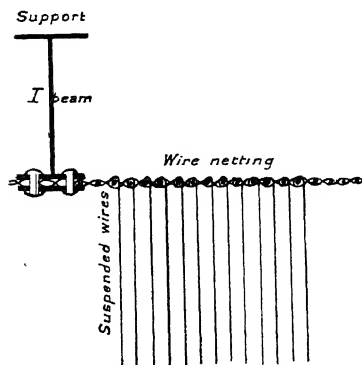


Fig. 214.—Rösing Wires.

**5. Filtering Methods.**—So far as the author is aware we owe the first suggestion of filtering methods to Percy,† but it was not until recently that Bartlett experimented on the subject with satisfactory results. The Lewis and Bartlett bag process has been already described in Chapter iv.

In considering the desirability of filtering blast-furnace gases, the fact should be taken into account that, whereas ordinary flue-dust can be settled with comparative ease by sufficiently

reducing the velocity of the gas current, there is always a certain amount of lead fume which leaves the furnace not as minute solid particles, but as true vapour, or rather as a cloud imperfectly condensed from the condition of vapour, and this cannot be completely settled out by any extent of flues, but only by filtering the gases through calico or flannel. This true fume, after all the dust has been settled out, may contain as much as 60 per cent. lead (chiefly as sulphide and sulphate) together with precious metals in amounts varying with the average assay of the furnace charge, but which may reach 8 to 15 ozs. silver, and, in the case of ores very rich in gold, from 0.05 to 0.3 oz. of that metal per ton. Generally speaking, though not invariably, the silver assay of the fume increases in proportion to the lead present with increased distance from the furnaces. As a preliminary to the filtering of the fumes, it is always advisable to settle out as much as possible of the fine ore-dust in ordinary flues and dust-chambers, which possess the further advantage of cooling the gases so that they do not

\* Saeger, *B.-u. H. Zeitung*, 1894, p. 299.

† *Metallurgy of Lead*, p. 449.

destroy the filtering fabric, the extreme limit of temperature admissible being about 120° C., and the desirable limit only 80° C.

For gases the sulphur contents of which are low, unbleached calico of 0·6 oz. per square foot can be used, with about 46 threads to the inch in both warp and woof. If, however, the gases are comparatively rich in SO<sub>2</sub>, the bags should be of unwashed woollen flannel, and when the gases are very rich in SO<sub>2</sub>, or contain SO<sub>3</sub>, as do those from roasting furnaces, no material will long withstand their effects. A common size of bag is from 18 to 20 inches diameter and 25 to 35 feet long, the material is lap-jointed and double sewn; the thimbles over which the lower ends of the bags are tied are generally of No. 10 sheet steel, the edge being turned over a  $\frac{1}{4}$ -inch wire, and they are spaced about 27 inches apart, the bags being tied on by a 2-inch strip of calico. The area of cloth required per ton of ore smelted varies from 300 square feet (Bingham Junction) to 750 square feet (Globe).

The bag-house of the *Globe*\* Works (Denver) is similar in general arrangement to those of the Lewis and Bartlett process shown in Fig. 32, except that instead of hoppers below the sheet steel floor carrying the thimbles or nipples there are only open brick-paved chambers called fume-rooms in three rows separated from each other by brick walls and divided up transversely also by brick walls, these precautions being necessary in order to prevent fires from spreading. The nipples are of sheet steel No. 8 gauge, braced by  $\frac{1}{4}$ -inch rods, and the bags attached to them by means of 2-inch strips of calico † wound round four times, are 36 inches diameter and 30 feet long, made of unbleached calico weighing 0·6 oz. per square foot, with 42 to 46 threads to the inch in both warp and woof, and costing about 2d. to 2½d. per yard, 62 inches wide. This fabric resists fairly well a temperature of 125° C. without loss of strength, but commences to char and becomes much weakened at temperatures of 150° and upwards; care is, therefore, necessary to avoid excessive heat arising from spontaneous combustion or otherwise, for which purpose a bye-pass current of cold air is provided separately for each room, which is opened whenever the temperature rises above 150°, the usual ignition point of the finely-divided fume being about 175° C. The filtering area provided was at first only 250 square feet per ton of ore smelted, but this Iles regards as an absolute minimum, and it has been increased to 750 square feet per ton. At these works not only the blast-furnace gases, but also the whole of the fumes from the roasters, and even those from the lead wells and slag taps, drawn off through an overhead main with funnel-shaped ventilators, are sent to the bag-house. The resulting fume is ignited on the floor of the fume-rooms, which are for that purpose isolated in turn from the current of gases, and the product is a porous grey cinder containing as oxides and sulphates Pb 75 per cent., Zn 3 per cent., Fe 0·5 per cent., As 1·3 per cent., and Ag 4 ozs. per ton. As regards the vacuum, Iles finds that 0·35 oz. at the fans is a minimum to prevent leakage at the charging doors of the furnaces, and prefers at least 0·5 oz.

At the *Omaha* Works ‡ (Neb.) the *Brown-De-Camp* filtering process has been in successful use since 1894, the gases from ten blast furnaces, each 42 by 120 inches at tuyeres, being first passed through 800 feet of brick

\* Iles, *Lead Smelting*, pp. 186-210, and *Private Notes*, 1897.

† It may be remarked that in the United States calico is termed "muslin."

‡ *E. and M. J.*, Oct. 19, 1895.

flues in order to settle out the dust, and then sent to the filtering house. Instead of bags, high-peaked ceilings are employed, each 29 feet high by 99 long, any one of which can be cut out of the series, and the material employed instead of woollen or cotton is a heavy, but open, woven canvas or duck. Each compartment of the filter has a series of ceilings, the effective filtering area of which is 40,000 square feet, supported by a framework of I-beams; the capacity of each compartment below the ceilings is 60,000 cu feet. It is found that, to ensure complete efficiency, discharging the gas quite colourless and without visible vapour, about 16,000 square feet are required for each furnace, the whole installation being thus sufficient for 10 furnaces. The fan is a 12 feet Murphy ventilator, which, at 200 revolutions, takes the fumes from ten furnaces through a flue of 64 square feet area, and consumes, roughly, about 7 H.P. for each furnace running. Suspended between the converging walls of the peaked ceilings are oscillating shafts carrying long beaters, which can be put in motion by means of levers from the outside of the building as often as the pressure gauge in each compartment shows that the meshes of the fabric are becoming clogged, causing the beaters to strike the canvas and shake off the dust, which falls to the floor of the building.

When the dust has accumulated to a sufficient extent it can be burnt on the floor, a separate set of flues and a small chimney being provided to carry off the products of combustion. It is usually, however, wheeled to a separate closed chamber for ignition, which converts it into a grey or pinkish friable coke, suitable either for the blast furnaces or for working up in the refinery together with skimmings and drosses to a hard lead.

The actual saving when running all the gases from seven furnaces through two compartments of the above condenser averages about 100 tons of metallic lead per month, carrying about 10 ozs. silver per ton, which is 6 per cent. of all the lead charged into the furnaces; this yields a gross income of about £1,200 per month, which is said to be much more than the total expense of maintenance, depreciation, and interest.

At *Murray*\* (Utah) the bag-house is a building 216 feet  $\times$  100 feet, and contains 4,160 bags, each 30 feet long  $\times$  18 inches diameter, grouped in five parallel sections, each of which ends in a steel chimney 175 feet high. The fan provided, which drives the gases, cooled by passage through long flues to a temperature of 120° to 160° F., through the bag-house, is 18 feet diameter by 6 feet wide, and delivers 250,000 cubic feet per minute under a pressure of 1 inch water, taking 125 H.P. The cost of the installation was £30,000.

At the *Bingham Junction*† Works of the U.S. Smelting and Refining Co. there is a similar bag-house in three sections with a steel floor 12 feet above ground level, provided with 2,200 nipples, 18 inches diameter, 6 inches high and 2 feet apart, to which are attached the usual calico bags, 32 feet long. The fan delivers against a pressure of 1 inch water, and takes 90 H.P. The total area of the 2,200 bags is 310,860 square feet, which, as they serve six furnaces, is 50,000 square feet per furnace, or 300 square feet per ton of metal seeing that the area of each bag is about 141 square feet, this amounts to just over two bags for each ton smelted.

At *Pertusola*‡ a filtering plant has been recently installed at the foot

\* *E. and M. J.*, Sept. 28, 1907, p. 576.

† *E. and M. J.*, Sept. 21, 1907, p. 531.

‡ *Mining Journal*, May 18, 1907, p. 665.

the chimney and at the end of flues, which are 1,800 metres in length. From ten to twelve filters are in use, each having a surface of 40 to 45 square metres, and about 500 cm. of gases are passed through per minute by the aid of Fiechter fans taking 10 H.P. The temperature of the gases is 70° to 80° C., and the fume collected carries 75 per cent. lead.

At the *Selby Works* (Cal.)\* a bag-house of re-inforced concrete has been recently erected for three blast furnaces of  $36 \times 144$  inches, smelting together about 540 tons per day, besides other accessory furnaces. There are 1,920 bags, 18 inches diameter and 28 feet 8 inches long, giving 135.1 square feet per bag, or a total of 259,392 square feet. The total allowance made was 410 square feet per ton smelted + 17 per cent. for other furnaces, or 480 square feet per ton in all. The fan is 11 feet diameter  $\times$  5 feet 6 inches wide, and at a speed of 116 r.p.m., taking 24.75 H.P., against a pressure of  $\frac{3}{4}$  oz. delivers 76,000 cubic feet per minute at a temperature of 112° to 114° F., which, at this works, is found to be best for the bags. According to Bennett,† the basis of bag area for a given tonnage smelted is unreliable, and it is better to reckon in terms of square feet per cubic foot per minute of gases passed through. At present the provision is 3.413 square feet per cubic foot of total gases entering the chamber, but, eliminating the accessory furnaces, which would be better served by a separate bag-house, and by dividing up the blast-furnace house into parallel sections, either of which could be isolated at will for cleaning purposes, the actual area of filtering surface required to be in use per cubic foot of gas per minute is only 1.14 square feet.

Even in a large plant the cost for filtering the blast-furnace fumes amounts to from 4d. to 5d. per ton of ore smelted,‡ so that many metallurgists are still doubtful if the capital outlay is really warranted by the results, in spite of the fact that the recovery of lead is increased by something like 1 per cent. per ton of ore smelted.

Until recently it was considered impossible to filter the fumes from roasting furnaces except when largely diluted with ordinary blast-furnace gases, on account of the high proportion of  $\text{SO}_3$  present, which rapidly corroded all the fabrics successfully employed for blast-furnace fume; yet the roaster fumes are far richer in lead, and, therefore, better worth filtering. According to a recently introduced improvement, referred to on a later page, the roaster fumes are neutralised by blowing in fumes of  $\text{ZnO}$ , after which they can be safely filtered through woollen bags, and the residues can then be leached for  $\text{ZnSO}_4$ .

6. Use of Electrical Discharges.§—Since the discoveries of Lodge and Clark in 1884, it has been known in a general way that discharges of static electricity would rapidly cause all dust to settle in a closed chamber, and patents were taken out by the firm of Walker, Parker & Co. in 1885 for the purpose of utilising these discoveries practically. Until recently, however, no success had attended experiments in this direction, because electrification of the dust particles can only take place in stagnant air; as soon as there is anything like a swift current of gases the discharge seems to be dissipated.

\* Bennett, *E. and M. J.*, Sept. 5, 1908, p. 456, where drawings of the plant are given.

† *Loc. cit.*

‡ *Iles, Lead Smelting*, p. 210.

§ Hutchings, *B.-u. H. Zeitung*, 1885, p. 253; *Iles, S. of M.Q.*, vol. xvii., pp. 119, *et seq.*

Considerable success is, however, stated to have been attained by the Cottrell process at the Selby Works (Cal.). No details are available as to the potential employed, but the following is condensed from the summary of the specification of the Cottrell patents.\* The conditions for successful deposition of suspended particles seem to be the following :—

1. The use of continuous currents.
2. The charging of suspended particles by means of electrodes with a large surface, over which a uniform discharge of the brush or glow type is maintained.
3. The attraction of charged particles by electrode surfaces of opposite polarity.
4. The passage of gases through the electric field in such wise as to secure fairly uniform treatment of the whole bulk of gas.
5. The maintenance of as perfect insulation as possible between the electrode surfaces of opposite polarity, more especially guarding against short circuiting through the material deposited out of the gases.

The charging electrode may take the form of a suspended wire cage wound with asbestos cord or thread, the projecting fibres of which form myriads of points for the glow discharge. The other electrode may be the walls of the flue containing the furnace gases. The connections and supports for the inner electrode must, of course, pass through the outer walls at certain points, and in order to guard against the deposition of a dust layer upon solid insulators, it is necessary either to force in at each of such points a current of pure dry air free from dust, so forming an insulating layer, or else to heat up the air in the neighbourhood of the insulator, by means of a heating electric coil or otherwise, to such a temperature that no dust can deposit upon it.

**Wet Condensation.**†—Many attempts have been made to employ water for condensing lead fumes, but the great difficulty is that of getting contact between the water and the minute particles of fume, which, instead of being attracted, are repelled by a moist surface, as first noticed by Percy.‡ Three main varieties of wet process may be described :—

1. **Passing through Water.**—This, although effective, is costly, owing to the power absorbed by the fans required to force the fumes through the head of water. Simple bubbling through water under a diaphragm or through perforated pipes was soon found to be almost useless, but the method of the *Sheffield Smelting Co.* is very effective. The fumes are distributed in a closed tank by means of a network of perforated pipes 7 inches below the surface, and above the pipes are fixed several diaphragms of copper wire cloth about 16-mesh, the action of which is to break up the bubbles and so increase the contacts of dust-laden gases with the water. The gases are forced along by means of Roots' blowers, and it is stated that the cost of the power required, although considerable, is much more than repaid by the value of the fume recovered.

2. **Blowing in Steam.**—This is cheap enough, but all attempts to utilise steam exclusively have failed in practice. Most of the steam escapes condensation, and the small portion condensed forms a strongly acid mud, which rapidly attacks the materials of the flues and chambers. That portion of the steam which escapes condensation hinders instead of assisting

\* *E. and M. J.*, Aug. 22, 1908, p. 375.

† *v. Percy, Metallurgy of Lead*, pp. 441-449.

‡ *Metallurgy of Lead*, p. 450.

the segregation of the dust particles, on account of the additional heat imparted, which causes the particles to repel each other.

**3. Passing through a Water Spray.**—This process, though costly in plant, has given good results at several works. At the *St. Louis Smelting and Refining Works* the plant consisted of a chamber  $20 \times 10 \times 12$  feet high, divided into three compartments, built of tiles, in sections containing shelves composed of drain tiles supported by planks borne on old iron rails. The smoke passed first up through one such tower and then down through another, being sprayed all the time through fine rose nozzles at the top of each tower. It was then driven through a long wooden tank containing water, and provided with a number of hanging curtains which forced the gases to come in contact with the surface of the water while they were again sprayed from above. The fans were two Sturtevant's, 7 feet 6 inches diameter, and the fume was perfectly condensed, but whether or not the process was a financial success is doubtful. In this case, as in that of the Sheffield Smelting Co. previously referred to, the works being situated in a populous city, perfect condensation was imperative regardless of cost.

At the *Bimetallic Smelter* \* (Leadville), now idle, a similar plant was formerly at work, consisting of two circular towers, each 20 feet high by 16 feet diameter, through which the fumes from two large pyritic furnaces were forced by means of two Huyett & Smith fans, 10 feet diameter, running at 260 revolutions per minute. The towers contained a great number of A-shaped tiles with serrated lower edges, supported in the form of shelves on old rails completely enclosed in sheet lead, the seams of which were "burnt" together, as were also those of the sheet-lead lining of the towers. Water was supplied from a shallow tank, forming the roof of the tower, through a network of 120 leaden pipes, and trickled down from shelf to shelf, falling from the serrated edges of one tile on to the ridges of those in the next row. The fumes entered the towers at a temperature of  $350^{\circ}$  to  $400^{\circ}$ , and left at a temperature of  $90^{\circ}$ , being discharged through an octagonal stack 150 feet high built of planks heavily tarred. The condensed fume was washed out by the stream of water issuing from the bottom of the towers, settled in tanks, and sun-dried. It contained 25 to 40 per cent. Pb and about 25 ozs. Ag per ton.

At *Monteponi* † the blast furnace and other fumes are passed through a water-spray near the commencement of the condensing system, and, therefore, while still quite hot. The condensing plant consists of an arched chamber, 20 feet long by 10 feet wide, divided into twelve compartments by means of one longitudinal and five transverse baffle walls, the arrangement being shown in Figs. 215, 216, and 217.

Each time the current of gases changes its direction and passes over the longitudinal partition, which it does six times, it is struck by a fine spray of water from a rose, the six roses discharging together about  $1\frac{1}{2}$  litres per minute. Of this total quantity of water, about four-fifths is vaporised, while one-fifth flows off through the water seals at the bottom of the condensing chamber, carrying with it from 10 to 15 per cent. of the total quantity of flue-dust produced in the plant. Settlement of the remainder of the flue-

\* Private Notes, 1897.

† Ferraris, *Oest. Zeit. f. Berg- und Hüttenwesen*, 1905, p. 455; abstract in *E. and M. J.*, Oct. 25, 1905, p. 783.

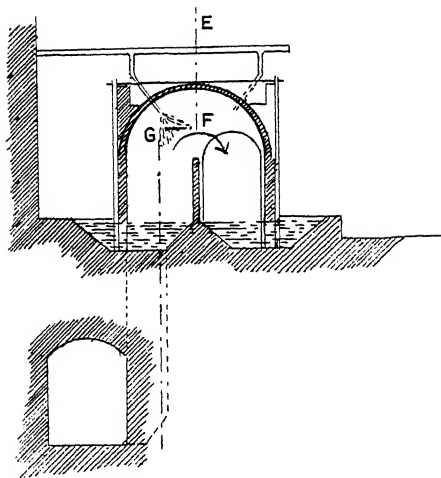


Fig. 215.—Section C D.

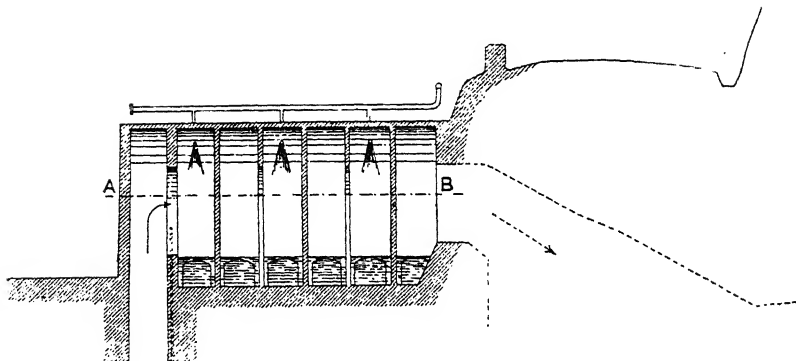


Fig. 216.—Section E F G H.

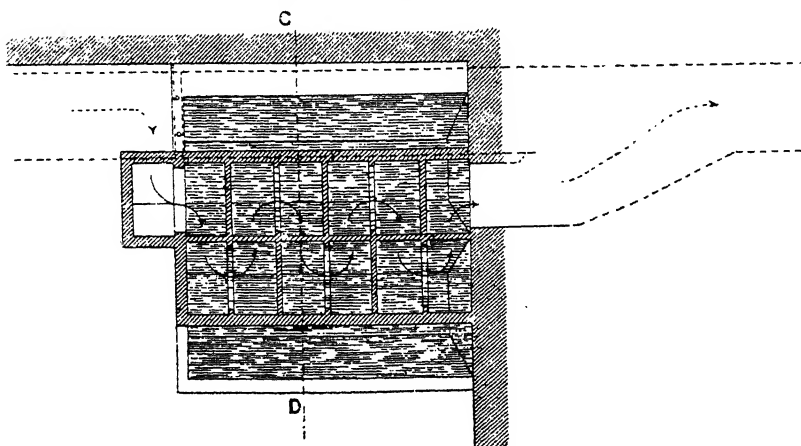


Fig. 217.—Section A B.

Figs. 215 to 217.—Water-spray Condenser (Monteponi).

dust in the long flues which follow is found to be much facilitated by condensation of the water vaporised in the condensing chamber, so that practically nothing escapes from the stack except a little steam. No fan or other artificial draught is required, but it may be remarked that the top of the stack at these works is no less than 524 feet above the level of the furnace floor.

At *Brimsdown* \* the fumes from eight cupellation furnaces are drawn off at a temperature of from 250° to 300° through high-speed fans, upon the blades of which jets of water impinge; the condensed fume is carried out of the fan casing by the stream of water and collected in settling tanks.

At the *Tarnowitz* Works a large fume-condensation plant has been recently installed, in which fans are employed to exhaust the gases from the furnaces, and drive them through a series of water-sprayed towers in which the flue-dust is collected.

At *Scheriau* † (Carinthia) the gases from the ore hearths and roasting furnaces together enter a sheet-lead flue, 8½ feet square and 180 feet long, and thence pass to the wet-condensation plant, which consists of 52 chambers, 15·7 feet square and 17·7 feet high, internally sprayed from the top, through which the gases circulate, being cooled during their passage down to 20° C., and having their sulphur content reduced from 13·8 down to 2·6 grammes per cubic metre. Beyond the wet-condensation plant is a stack 98 feet high, but this would be insufficient to handle the cooled gases without the insertion of a suction fan making 720 revolutions per minute, and taking 35·3 cubic feet of gas per revolution.

**Quantity of Flue-Dust Recoverable.**—The quantity of flue-dust produced is influenced by various factors, among which may be mentioned the amount of fine ore in the charge, degree of friability of the fuel, pressure of blast employed, amount of bosh in the furnace shaft, care in keeping the walls free from accretions and the top cool, and, especially, the height of the column of charge in the furnace.

With filtering methods, practically the whole of the lead volatilised can be collected as fume, but with all the other methods a variable amount always escapes, in many works little else but ore dust being collected.

At the *Keld Head* smelting works in 1857 ‡ the fume, collected by means of water-spray condensation from ores which yielded in the ore hearth 1,374 tons of lead, gave 96 tons of fume lead, which is 7 per cent. of the ore yield, or about 5 per cent. of the total weight of ore charged, and over 6 per cent. of the weight of lead present.

Hofman § puts the average quantity of flue-dust *formed* in American works at 5 per cent. of the total weight of ore charged, which is probably an extreme figure for high modern furnaces; the quantity *collected*, however, averages barely half as much.

According to Pufahl, || the following figures show the percentage proportion borne by the flue-dust collected to the ore smelted at a number of the leading American works :—Globe (Denver) 0·5 per cent., Eilers (Pueblo) 0·5 per cent., Pueblo (Pueblo) 0·9 per cent., East Helena (Mont.) 0·3 per cent., Arkansas Valley (Leadville) 0·2 per cent., Murray (Utah) 1·2 per cent.

\* *Private communication*, 1908.

† *Min. Ind.*, vol. xv., p. 536.

‡ *Perey, op. cit.*, p. 456.

§ *Hofman, Metallurgy of Lead*, 8th ed., 1906, p. 380.

|| Quoted in *Mineral Industry*, vol. xiv., p. 399.



The general average of all these figures is 0.6 per cent., which, however, is probably below the truth.

At the *Smelter No. 3, Monterey*,\* the flue-dust saved amounts to 1.2 per cent. of the weight of ore charged, and from 1 to  $1\frac{1}{2}$  per cent. on the ore smelted will be an average figure for other plants in Mexico.

At Ems † in ten months of 1881-82, the 14,605 tons of ore treated, containing 42.8 per cent. lead, yielded 931 tons of flue-dust, averaging 56.9 per cent. of lead; the flue-dust actually collected, therefore, was 6.4 per cent. by weight of the ore treated, and contained 8.5 per cent. of the lead contained in it. At Freiberg ‡ in 1893 the weight of fume actually collected amounted to no less than 9.7 per cent. of the weight of ore smelted, and contained 1,658 tons lead, 36,655 ozs. silver, and 917 tons of white arsenic, the total value being £13,636.

It has been already mentioned that at Omaha (with the new appliances) the fume collected, as an extra saving over and above the ordinary flue- and chamber-dust collected in 800 feet of flue, is equivalent to 6 per cent. of the lead charged into the furnace. It may, perhaps, be reckoned that, as a rule, in works smelting ores fairly rich in silver, from 2 to 3 per cent. of both lead and silver will be carried away mechanically as fine dust, which can all be settled out readily in suitable chambers; while from 5 to 8 per cent. of the lead, and  $\frac{1}{4}$  to  $\frac{1}{2}$  per cent. of the silver, will be volatilised, more or less of which is condensable according to the degree of perfection of the appliances employed. With ores poorer in silver, the quantity of that metal volatilised is much greater in proportion to the lead, and may reach, or even exceed, 1 per cent. of the total amount present.

**Furnace Gases.**—The deleterious effect of sulphurous gases upon vegetation has caused a good deal of attention to be directed lately to the neutralisation of furnace gases, particularly those from roasting furnaces, which contain the highest percentage of sulphur, and are, therefore, the most injurious in their effect.

At the *Bingham Junction Works* of the U.S. Smelting and Refining Co. a method has been recently introduced of neutralising the sulphurous gases by means of finely-divided zinc oxide produced in small "blowing up" furnaces, 6 feet long by 3 feet wide, provided with a perforated grate upon which is fed a dampened mixture of zinc sulphide concentrates with 50 per cent. of its weight of fine coke, the depth of the bed of the material being maintained at about 12 inches. Each of these furnaces handles about 5 tons of material per day, and the resulting zinc oxide fumes mingling in the flues with the sulphurous gases from the roasting furnaces form zinc sulphate, which is collected in woollen bags, of which there are 2,300, each 18 inches diameter by 33 feet long. These are shaken every twelve hours, and the dust falls into a cellar, where it is burned before being drawn off into cars.

#### TREATMENT OF FLUE-DUST.

The methods adopted for treating flue-dust have nothing to do with the mode of collection. The use of lead fume in its natural condition as paint is more than two centuries old, and at Linares and other places large quantities are sold for use as such, but the grey colour is frequently objectionable.

\* *Private Notes*, 1902.

† Egleston, *Trans. A.I.M.E.*, vol. xi., p. 404.

‡ Bauer, *Jahrbuch f. d. B. u. H. W. im K. Sachsen*, 1894, p. 39.

The special treatment of non-argentiferous fume from ore hearths to produce a white pigment has been described in Chapter iv., but with ordinary fume and flue-dust the object is simply to return it to the ore furnace, so as to obtain its lead and silver contents in the form of metal.

Formerly the commonest way of handling flue-dust was to wet it down, forming a kind of mortar, which, after partial air drying, were fed back into the furnace. Flue-dust from roasting furnaces, which commonly contains a good percentage of zinc sulphate, sets fairly hard when moistened, but the ordinary flue-dust has little coherence unless previously ignited. At the *Broken Hill* plants, however, where lead is cheap and fuel and labour is dear, this was, until quite recently, the only method adopted for handling the flue-dust, although much of it was quite obviously blown out again by the blast unaltered. Nowadays, however, flue-dust is always either *fused* in some form of reverberatory furnace, or *bricked*.

1. *Fusion*.—Where slag- or sinter-roasting was carried on it was a common practice to add a few cwts. of flue-dust to the slagging hearth immediately before dropping the hot-roasted charge. In this way the mechanical loss was small, but the volatilisation loss of lead was heavy.

At *Mazzaron* (Spain) the flue-dust is bricked with fine galena, lime, and a little sand, before being added to the charge on the last hearth of the reverberatory roasting furnace; the object being to still further diminish the mechanical loss in charging and mixing.

At the *Globe Works* a small separate reverberatory is used for melting down flue-dust, the fumes being all collected and filtered in the bag-house. A very good slagged product is made, but the loss of lead (25 per cent.) would be prohibitive, were it not for the advantage of being able to save it all by filtering.

2. *Bricking*.—This is by far the most common method of handling flue-dust, and fine ore and roasted matte are frequently bricked together with the flue-dust. The principal binding material employed is slaked *lime* in the proportion of 2 to 10 per cent., which is mixed with the flue-dust in a pug-mill and bricked either by hand (in which case they are tender) or by machine. *Clay* as a binding material makes stronger bricks than lime, but is objectionable on account of introducing extra siliceous material into the charge. At *El Paso*, 5 per cent. of clay was formerly mixed with the flue-dust in a pug-mill, and the bricks were made by hand and air-dried; they stood a fair amount of handling, but the cost was 12s. per 1,000 bricks of ordinary size. *Pan slimes* were used instead of clay at Tombstone (Ariz.) by Church.\*

Among other binding materials may be mentioned *green vitriol*, used by Hahn (the objection to which besides its cost is the addition of sulphur to the charge) and *dilute sulphuric acid* used by Eurich, which acts by the formation of basic sulphates and their "setting." *Sugar waste* (coarse molasses) has been used at Deadwood by Carpenter,† who found that it gave good results, besides improving reduction.

At *Laurium* (Greece) ‡ the flue-dust produced, which contains 40 per cent. Pb and 15 per cent. As, is mixed with slimes from the dressing works, and screened raw ore-fines, in heaps of 60 tons and about 4 feet in height

\* *Trans. A.I.M.E.*, vol. xv., p. 611.

† *Ibid.*, vol. xxx., p. 1137.

‡ *Private Notes*, 1904.

These are watered with a hose, and shovelled into soft-mud auger-type brick-machines with two nozzles,  $6 \times 4$  inches, the bars being cut by wires, so as to make bricks of 22 lbs. each, which, after air-drying for two days, are stacked in heaps.

At *Goslar* the flue-dust is rich in zinc oxide, which is first leached out with dilute sulphuric acid, and recovered from the liquors as white vitriol, while the residue containing the lead and silver is bricked for smelting.

Much stronger bricks can be made with a smaller proportion of binding material when machines which exert a powerful pressure are employed. There are two chief types of machine—the *Chisholm, Boyd & White Mineral Press*,\* in which a heavy roller presses the slightly damp mixture into hollows formed in a revolving mould wheel, which are successively presented under its periphery, and the *White and Boyd* type presses, in which plungers are driven into cylindrical moulds by toggle joints or cams, or by a combination of both, the pressure frequently amounting to as much as 2 tons per square inch. Both of these types of machine have been described in Chapter ix.

At the *Germania Works*† (Salt Lake, Utah) the flue-dust is mixed in a slightly damp condition with enough slaked lime to make 5 per cent.  $\text{CaO}$  in the final brick, and is then fed into the hopper of a Mineral Press, which forms flat circular cakes about 4 inches diameter, weighing 3 lbs. each, at the rate of 4,200 per hour, equivalent to 55 tons in a 10-hour shift. These can be used in the furnace after only two days air-drying, but are better left for a week.

At the *Selby Works* (San Francisco) the briquetting plant for flue-dust and fine ore consists of a 7-foot Chilian mill for grinding and mixing, and a Boyd four-mould brick press, the total capacity being 75 tons in ten hours.

At *Pueblo* (Colo.) the flue-dust is mixed with 5 per cent. of lime, and briquetted in a six-mould White press.

In all cases the strength of bricks containing a considerable proportion of flue-dust can be increased by burning or firing.

At *Smelter No. 3, Monterey*,‡ the flue-dust is mixed with about 15 per cent. of clay and bricked by hand at a cost of only 3s. per 1,000 bricks, but in order to give the bricks sufficient strength they are stacked in an ordinary brick kiln and lightly burned, in which partial roasting a considerable quantity of  $\text{SO}_2$  and  $\text{As}_2\text{O}_3$  are eliminated. The composition of the burnt bricks is given in Table XXXVIII.

At the *Broken Hill Block 14 Works*,§ Greenway briquetted flue-dust with fine ore in a semi-dry press after pugging with milk of lime in a horizontal pug-mill, and then stacked the bricks in kilns of 30 to 50 tons capacity, and fired them during three to six days. The total cost of the operation was from 4s. to 6s. per ton of material, and the bricks were very strong.

\* See Figs. 193 and 194, Chap. ix.

† *Private Notes*, 1902.

‡ *Private Notes*, 1897.

§ *E. and M. J.*, 1905, vol. lxxix., p. 73.

## SECTION III.—DESILVERISATION.

### INTRODUCTORY.

THE distinction between argentiferous and non-argentiferous lead ores has been already emphasised (Chapter ii.); as also the fact that the latter are usually much purer and more free from base metals. Thus the lead obtained from Missouri and Wisconsin lead ores (as also a part of that from Spain and Germany) is comparatively free from silver, as well as from base metals, and can, therefore, be manufactured almost as it leaves the smelting furnace after a simple "poling," or at most a few minutes "blowing" with a steam jet followed by drossing. All the lead produced in blast furnaces from complex ores, however, as well as a large part of that produced in reverberatories, contains at least sufficient silver to pay for its extraction, even if, indeed, the silver contents be not more valuable than the lead itself, as in the case of Freiberg, Colorado, and Broken Hill ores.

**Classification of Methods.**—The desilverisation of lead bullion may be performed by three different methods—viz., 1, Cupellation; 2, the Pattinson process; and 3, the Parkes process. The first method, though the simplest, is the most expensive for labour and the most wasteful of material; yet it may still be adopted with advantage, in spite of its high losses, for the treatment of rich lead in very inaccessible districts where silver is really the valuable metal sought, and lead (owing to the heavy freight) almost valueless. Such conditions still prevail in a few parts of Mexico and in the mountainous districts of S. America and elsewhere. In such cases it is sometimes advisable to adopt a direct cupellation process, which turns out the silver at once in a portable form, while the litharge is either reduced to metal in an open hearth or simple furnace, or, more commonly, is utilised as a source of lead in smelting silver ores.

Direct cupellation has the advantage that it can be carried out, if necessary, with no other material than clay or sun-dried bricks, no other fuel than brushwood, and no other blowing machinery than a common blacksmith's bellows. For further information as to simple cupellation hearths, two papers by Austin,\* and one by the author† may be consulted with advantage.

In all cases, however, where fuel is cheap and the country moderately accessible by means of railways or good roads, the direct cupellation process should be replaced by either the Pattinson or the Parkes process. An example of a case where this was not done at the proper time is afforded by

\* *Trans. A.I.M.E.*, vol. xii., p. 41, and vol. xiii., p. 185.

† *Trans. I.M.M.*, vol. xii., p. 121.

the Las Trojes smelting works in Michoacan, Mexico,\* which are no longer in operation as lead smelting works. Hofman quotes from Reich the following table, which shows at a glance the comparative disadvantages of cupellation as compared with the Pattinson and Parkes processes, and is, therefore, worth reproduction. The figures are from the Ems works, where all three processes were carried out successively.

	Cupellation.	Pattinson.	Parkes.
Cost of treatment per ton, units, . . . .	3	1½	1
Amount of lead to be cupelled, units, . . .	100	13	5
Percentage recovery of lead and silver, . .	94	98	99
Silver lost in the lead, ozs. per ton, . . .	0·73	0·58	0·17
Traces of gold, . . . . .	lost	lost	recovered

Electrolytic processes are referred to in a separate chapter. Only one of the many methods tried experimentally has been commercially successful, and as yet it has only been adopted in two or three works.

Practically speaking, therefore, the desilverisation of lead upon a large scale is almost invariably carried out by either the Pattinson or the Parkes process. Both are English processes, and both have been brought to their highest state of perfection in this country, but unfortunately the traditions of English smelters and refiners have always led them to keep their operations as secret as possible, even in cases where their foreign competitors have already copied most of the features that are worthy of imitation.† Few particulars, therefore, are available as to English practice, and the descriptions of plant, &c., must be mostly drawn from foreign localities.

**Influence of Impurities on Desilverisation.**—Almost all the impurities in the work-lead interfere, to a greater or less extent, with desilverisation by either process, although, generally speaking, the Parkes process requires a purer lead than the Pattinson.

The most harmful impurity is *arsenic*, which, even in small proportion, is fatal to good results; *antimony* also, even in small quantity, interferes greatly with the Pattinson process, and renders good work in the Parkes process impossible. Next in order among harmful impurities come *tin*, *zinc*, *sulphur*, *iron*, *nickel*, and *cobalt*. Fortunately, all the above have a greater affinity than lead for oxygen, and can, therefore, be eliminated by simple oxidation at a low red heat and skimming the molten bath.

*Copper*, which is, perhaps, as common an impurity as any of the above, has much less affinity for oxygen and is only imperfectly got rid of by oxidising influences. It forms, however, with part of the lead itself an alloy which is much less fusible than the remaining and purer lead, and, therefore, the bulk of it can be readily eliminated by means of a process of liquation or slow melting down, a smaller proportion, by subsequent cooling, gradually

\* *Proc. Inst. Civ. Eng.*, vol. cxii., pp. 146, *et seq.*; 165, *et seq.*

† An amusing reference to this condition of affairs is to be found in the concluding remarks of the presidential address of Dr. R. Pearce (himself an Englishman and a pupil of Dr. Percy) to the American Inst. of Mining Engineers in June, 1889. *Trans. A.I.M.E.*, vol. xviii., p. 72.

rising to the top as a second crust or dross, which can be skimmed off. The first dross, besides copper, contains also most of the metals of the iron group contained in the lead, as well as some arsenic, sulphur, and tellurium, if present.\* Copper is not specially harmful in the Pattinson process, but greatly interferes with the Parkes process, increasing the time and the consumption of zinc.

*Bismuth* has distinctly less affinity for oxygen than lead itself, and cannot be readily separated from it; fortunately, this metal does not interfere at all with desilverisation, even in the proportion of 2 per cent., as shown by the experience at Freiberg. *Zinc* is a rare impurity in furnace-lead, except in minute traces, but is contained in considerable quantity in lead which has been desilverised by the Parkes process, and must be got rid of by oxidation before the desilverised lead is marketable.

The process of desilverisation adopted must, therefore, be preceded by a process of oxidation to remove impurities, which is called "softening," or sometimes, in England, "improving"; while the Parkes process is followed by another similar oxidation to remove zinc, which is usually called "refining," or, sometimes, "dezincing." Both operations require similar plant, and may be conveniently described together in a single chapter. The concentration of precious metals in rich lead by the Pattinson and Parkes processes follows in two separate chapters, and, lastly, the cupellation of the rich lead and refining of the silver.

As an alternative to the combined softening and desilverisation by ordinary methods, Rösing † experimented at Tarnowitz (Silesia) on a Bessemerising process in a converter, with basic lining, taking charges of  $6\frac{1}{2}$  tons. By this means lead containing 12·4 ozs. silver was almost entirely converted into a fume containing 75 per cent. lead and 2·4 ozs. silver per ton, the residual rich lead carrying 1,863 ozs. to the ton. The author has not succeeded in learning any later particulars as to the practical working of this process, but the losses of lead and silver consequent on the re-treatment of so large a quantity of flue-dust would seem to be a fatal objection.

\* Tellurium is found in notable quantity in the dross from base bullion produced at Partusola (Italy). v. Heberlein, *Berg- u. Hüttenm. Zeitung*, 1895, p. 41.

† See *E. and M. J.*, April 16, 1892.

## CHAPTER XIV.

### SOFTENING AND REFINING FOR MARKET.

THE softening of lead as a preliminary to desilverisation may take place in cast-iron kettles or in reverberatory furnaces. The former are suitable for softening lead practically free from antimony and arsenic, and containing only small proportions of copper. The latter are much more commonly employed, since they are adapted to the softening of more impure lead, containing large quantities of copper, arsenic, and antimony. When lead contains  $\frac{3}{4}$  per cent. of copper and upwards, as at Freiberg and Przibram, it is sometimes considered advisable to preface the ordinary softening process by liquation in a special liquation furnace with inclined bed, the lead gradually draining out into a kettle, from which it can be tapped or syphoned into the softening furnace. The same object, however, can be attained by a very gradual melting down in the softening furnace. In any case, the rapidity with which softening proceeds depends on the exposure of fresh surfaces to oxidation.

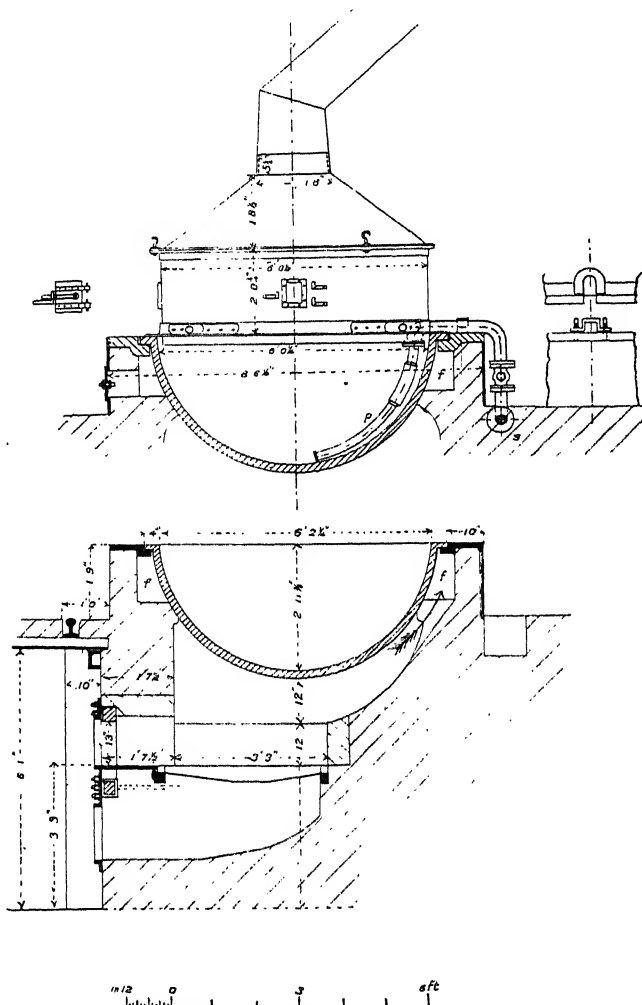
Refining, like softening, may be carried out either in kettles or in reverberatories. In the former, steam is the active agent (Cordurié's process); in the latter, atmospheric oxidation is exclusively relied upon, the process being slower and, perhaps, less thorough, but on the whole less costly.

**Softening in Kettles.**—At *Laurentthal* (Upper Harz) \* the softening, desilverisation, and subsequent refining are all conducted successively in the same kettle. Analyses of the work-lead treated are given in Table XXXIX.; it is comparatively free from arsenic and copper, and not very high in antimony. The kettles used are 6 feet  $2\frac{3}{4}$  inches (1.90 m.) diameter by 2 feet  $11\frac{3}{8}$  inches (0.90 m.) deep, and hold about 15 tons of lead each; their arrangement is shown in Figs. 218 and 219. The lower part of the kettle is heated directly by the flame, which then rises through the opening shown by the arrow and circulates round the upper part of the kettle in the flue, *f*, before passing, by a downcast, to the principal chimney flue. The hood and steam apparatus shown in Fig. 218 is not used for softening, but only for refining, for which purpose also a removable cast-iron pipe, *p*, hangs in the kettle and can be connected at will by means of a valve with the steam pipe, *s*.

**Softening.**—The charge of 15 tons of lead (containing 0.28 per cent. Cu and 0.57 per cent. Sb) is slowly melted down at a low temperature, taking six hours, at the end of which time two-thirds of the copper, together with all the sulphur and one-half of the arsenic, iron, and nickel, will have risen to the surface as a pasty "dross," which is skimmed off, leaving, however, practically all the antimony behind alloyed with the lead. A sample of the

\* Schnabel, *Handbuch der Metallhüttenkunde*, vol. i., pp. 451, et seq.

lead at this point poured into a mould shows, on cooling, a bright white frosted spot in the centre of its dull greyish-white surface, which, providing the lead has been poured at a low temperature, is an infallible indication of the presence of antimony or arsenic, and gradually disappears as the impurity is oxidised out.



Figs. 218 and 219.—Softening and Refining Kettle (Lautenthal).

The desilverisation with zinc follows, lasting fifteen hours and removing most of the remaining copper, without, however, affecting the antimony.

**Refining.**—The cooled desilverised lead, now containing 0·7 per cent. of zinc as well as the original 0·57 per cent. Sb, is then heated to a dull red by four hours' strong firing, the hood lowered to prevent access of air as far as



possible, and steam at 30 to 40 lbs. pressure is blown through the bath to oxidise the zinc. It is impossible to prevent the introduction of air along with the steam, and, therefore, some lead and antimony are oxidised together with the zinc. In kettle refining, however, the greatest care must be taken to use only dry steam, free from condensed water, otherwise dangerous explosions may occur; it is, therefore, a useful precaution to pass all the steam through a steam-trap if the pipe from the boilers is of any considerable length, and in any case to have a drain-cock in close proximity to the kettle. Steam must be turned on very gently to prevent the projection of some part of the charge.

Expulsion of the zinc generally takes about two hours, at the end of which time the hood is raised and the oxides skimmed off. The hood is again lowered, its four working doors are opened and steam is blown through with free access of air to oxidise the antimony, with formation of a thick semi-fluid mass of lead antimoniate. This also takes about two hours, when fluid yellow feather-litharge begins to form showing the antimony to be practically all expelled; after cooling, the lead is then cast into moulds by means of the Rössing lead pump.\*

The yellow powder of zinc and lead oxides is separated by hand budding from the shots of lead, and the most impalpable part of it, which floats away, forms, after drying, a good reddish-yellow paint containing 60 to 67 per cent.  $\text{ZnO}$ , the remainder being practically all  $\text{PbO}$ . In order to avoid so much oxidation of lead, Rössler† has forced in, together with the steam,  $\text{CO}_2$ , which yields a nearly pure zinc oxide; also a mixture of CO and nitrogen, which yields a considerable proportion of the zinc as "blue powder"—i.e., in the metallic condition. The improvement has not, however, come into general use.

The great drawback of kettle refining is the wear and tear of the kettles, which are rapidly corroded away at the surface of the metallic bath by the lead antimoniate. At *Lautenthal*, cast-iron kettles only last for 20 to 40 charges; at *Tarnowitz*, with a lead almost free from antimony, cast-iron kettles last for 90 charges; while those of open-hearth steel last 120 to 150 charges; and those of crucible steel 250 to 270 charges. Whatever their material, kettles for these and other purposes should always be cast bottom downwards, though the other way up is easier for the foundry; in practice, too, it is found that the best way of buying such kettles is with a guarantee that they will last so many charges, with a premium for each extra charge.

On account of the heavy wear and tear on the kettles, however, the method is gradually going out of use, *Pertusola* and *Port Pirie* among other places having followed the lead of English and American refineries. At the present moment it may be said that, apart from a few German works, kettle refining has been replaced almost everywhere by the more economical refining in reverberatories. The practice at the *Pueblo* plant will, however, be referred to on a later page.

**Liquation of Cupriferous Lead.**—At *Freiberg*‡ the work-lead containing 0·9 per cent. copper and other impurities (v. analysis in Table XXXIX.)

\* *Berg- und Hüttenmannische Zeitung*, 1889, p. 262; also Hofman, *Metallurgy of Lead*, 1906, p. 480.

† *Berg- u. Hüttenmannische Zeitung*, 1890, p. 248.

‡ *Schertel, Berg- u. Hüttenmannische Zeitung*, 1882, p. 293.

is liquated, together with all the skimmings from the subsequent Pattinsonisation, on the hearth of a special furnace very similar to that figured in Chapter xvi. The pigs are piled on an inclined hearth of marl, and gently heated by means of the flame from a closed grate supplied with blast under the bars, so as to burn inferior slack coal. The lead trickles out and collects in a pot, from which it is tapped from time to time through a spout. In twelve hours 12 tons of base work-lead are slowly melted down in the above furnace with a consumption of about 2 tons of slack coal; and from 4 to 5 per cent. of residues remain upon the hearth, containing practically all the sulphur, 93 per cent. of the copper, 96 per cent. of the nickel and cobalt, 77 per cent. of the iron, and 25 per cent. of the arsenic of the original lead, which, however, retains all its original bismuth, all its tin but 1 per cent., and all its antimony save about  $5\frac{1}{2}$  per cent. The dross melted down and allowed to cool separates into three separate products—viz., lead, speiss, and matte.\* The comparative absence of iron from the two latter, however, proves that they are not original furnace products held in suspension by the lead, but that the natural affinities of S for Cu and of As for Ni, Cu, and Pb come into operation to form compounds less fusible than the lead, by which they were dissolved in the smelting furnace at a much higher temperature.

TABLE XXXIX.—ANALYSES OF WORK-LEAD BEFORE AND AFTER LIQUATION.

Reference,	Lautenthal.		Freiberg.		
	Before Drossing.	After Drossing.	Before Drossing.	After Drossing.	Liquated Dross (5 %).
	1	2	3	4	5
Pb, . . . . .	98·96475	99·1883	96·667	97·830	62·40
Ag, . . . . .	0·1413	0·1440	0·549	0·536	0·17
Cu, . . . . .	0·2838	0·0907	0·940	0·042	17·97
Cd, . . . . .	tr.	none	..	..	..
Bi, . . . . .	0·0082	0·0083	0·066	0·066	none
Sb, . . . . .	0·5743	0·5554	0·820	0·771	0·98
As, . . . . .	0·0074	0·0032	0·449	0·333	2·32
Sn, . . . . .	none	none	0·212	0·210	0·04
Fe, . . . . .	0·0089	0·0048	0·027	0·006	0·43
Zn, . . . . .	0·0024	0·0015	0·022	0·021	0·07
Ni, . . . . .	0·0068	0·0038	} 0·055	0·001	1·09
Co, . . . . .	0·00035	tr.			
S, . . . . .	..	..	0·200	0·180	4·00
O, . . . . .	..	..	..	..	1·87
Slag, &c., . . . .	..	..	..	..	8·66

References.—1 and 2. Hampe, *Zeitschr. f. B. H. u. S. W. im Preussen*, vol. xviii, p. 203. 3 and 5. Schertel, *B.-u. H. Zeitung*, 1882, p. 293. 4. By difference from 3 and 5.

Neither at *Freiberg* nor at *Przibram* is the liquated lead sufficiently pure to be passed on to the Pattinson process without further softening; most.

\* *Ibid.*

modern works, however, treat lead less cupriferous than that mentioned above, so that it is possible to purify it sufficiently for desilverisation in a single furnace with great saving of time and fuel. Still, at *Port Pirie*, it has been found very advantageous to employ a system of two separate, but similar, furnaces for drossing and softening, as will be described hereafter.

**Softening in Reverberatories.**—The operation consists of two stages—viz., drossing and oxidation; the former is a liquation which takes place during melting down, the dross which rises to the surface containing most of the copper. The temperature is then raised to cherry red, and with free access of air tin, arsenic, and antimony oxidise successively, and are removed by skimming.

In Germany and at a majority of works in Europe, softening furnaces are still used, in which the hearth is only cooled by radiation through the foundation pan; this, formerly always of cast iron, is now usually made of wrought-iron plate.

At *Freiberg* the charge for the softening furnace is 21 tons, which takes thirty-six hours to soften completely with the aid of an air-blast, and yields 85 per cent. by weight of softened lead. Three kinds of skimmings are taken off, which are sometimes divided into six classes as one variety runs into the next. The first skimmings after the melting down are of a dark yellow colour, contain most of the tin, and are, on that account, powdery. They consist mostly of antimoniate of tin and stannate of lead, but as oxidation proceeds antimoniates and arseniates of lead are found and the skimmings become pasty. All these skimmings (*abstrich*) are drawn near the door nearest the flue and are called “tin skimmings.” Fumes of arsenic and antimony then appear, and the former element (together with some of the latter) oxidises rapidly to a light brown arseniate of lead, which is drawn off with a long iron hoe having a wooden blade and called “arsenic skimmings.” The brown arseniate of lead gradually gives place to the almost black antimoniate and the next lot of skimmings is called “antimony skimmings.” As the antimony becomes eliminated the black is more and more replaced by greenish-yellow litharge.

So long as there is any perceptible amount of antimony in the lead, a sample in a ladle will show globules of black skimmings revolving on the unoxidised surface of the metal like grease on water; but when nearly all the antimony is “out” the globules are replaced by a uniform yellow coating of litharge which forms quickly. Poured into a mould and skimmed just before solidification the bar shows the characteristic even, smooth, blue surface and lustre of soft lead.

At *Przibram* the charge is 22 tons, which is softened in twenty-four hours with a consumption of 9 per cent. of coal, and yields 81 per cent. of its weight as softened lead.

At modern works the wrought-iron pan is rivetted up to the sides of the furnace, forming an iron box which rests upon steel girders or heavy rails. Within this box the furnace walls are built up quite thin (9 to 13 inches), in order to cool as far as possible by radiation, and reduce the corrosion of the walls by lead, antimoniate, and oxide; and in many cases a water pipe or small water jacket is employed near the level of the top of the lead charge, where the side walls become most seriously affected. Most American furnaces are built within two complete pans, having a space between from 3 to 4 inches

wide, filled with water; but the cooling effect is then very great and causes excessive consumption of fuel to soften the lead sufficiently quickly. The

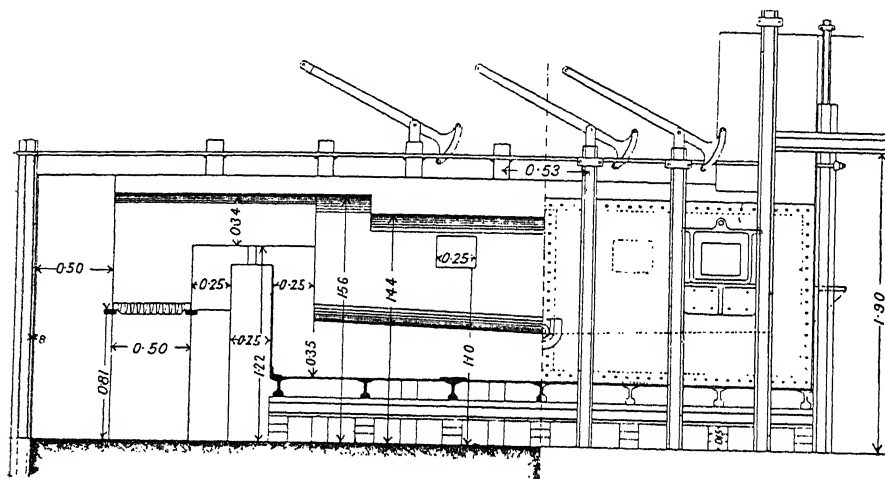


Fig. 220.—Softening Furnace, Monteponi. Side Elevation (part Sectional).

corrosion of the hearth is never serious and air cooling is quite sufficient; while the corrosion of the walls when water jacketed is very easily repaired with common clay\* or with brasque which is tamped in between charges.

#### FURNACE CONSTRUCTION.

The common type of softening furnace, which is a simple air-cooled box of iron plate, inside which the brick lining is built up, resting upon heavy rails or I-beams, is exemplified by Figs. 220 and 221, which show the furnace in use at *Monteponi* † (*Sardinia*). Similar furnaces are in use at *Peñarroya*, *Pertusola*, and other places, differing from that figured only in their capacity and in small details.

In England similar furnaces are often built of a much larger size,

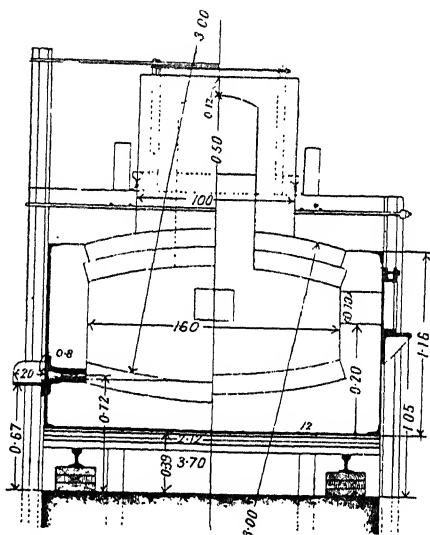
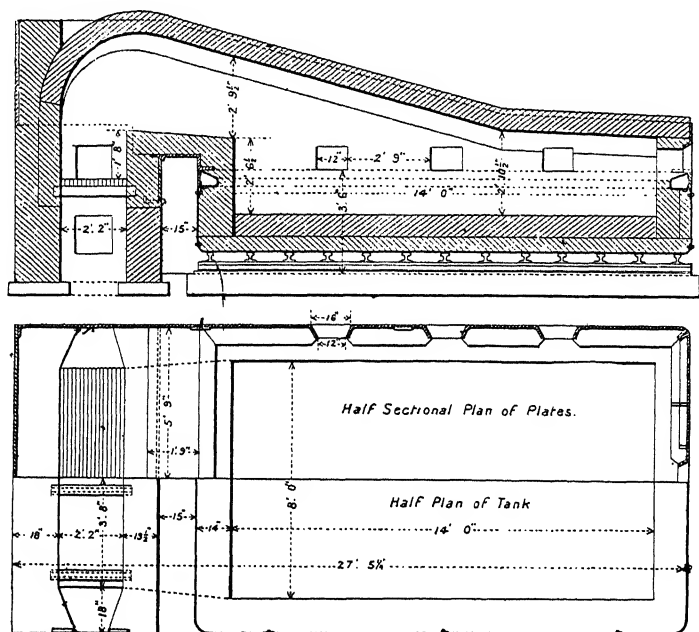


Fig. 221.—Softening Furnace, Monteponi (Transverse Section).

\* Common clay is much better than fireclay for almost all internal "patch" repairs of furnaces of every kind, ground firebrick, however, may be with advantage mixed in with it.  
 † *Ferraris, Oest. Zeit. f. B.- u. H W.*, 1905, p. 455; and *E. and M. J.*, Oct. 28, 1905, p. 781.

taking up to 140 tons of lead, and are sometimes provided with a water pipe extending round between the shell and the brickwork at the level of the lead bath.

The construction of a 36-ton water-jacketed softening furnace at the *Port Pirie Works* \* of the *Broken Hill Proprietary Co.*, divested of its binders and water connections for simplicity, is shown in Figs. 222 to 226, in which *a* is the hearth separated from the firebox, *b*, by an air space beneath the bridge, which is supported by a massive casting as shown. The hearth is wholly enclosed in a tank of  $\frac{5}{16}$ -inch mild-steel plate, to the upper part of which is attached the mild-steel water jacket of wedge section, detail of



Figs. 222 and 223.—Port Pirie Softening Furnace.

which is given in Fig. 225. The tank containing the hearth is supported upon a layer of 80-lb. rails pretty close together, as shown. The water jacket has  $\frac{1}{4}$ -inch water connections, and handholes here and there for cleaning. The position of the taphole is not shown, but Fig. 226 gives detail of the cast-steel tapping jacket, the taphole in which is formed by a screw plug having a hole through it  $1\frac{1}{8}$  inches diameter, which is itself closed by a wooden plug. In some of the furnaces (the so-called "copper furnaces") the tapping jacket is placed at the side, while in others (the so-called "antimony furnaces") it is placed at the end, its exact position being determined by the most convenient arrangement of the works. The whole of the furnace outside and above the steel-plate tank is enclosed in a close fitting casing of massive cast-iron plates  $\frac{3}{4}$ -inch thick, of which the door frames form an integral part; and the furnace is also strongly braced with old rails and tie-rods

\* *Private Notes*, 1896; see also Bayly, *Trans. Aust. I.M.E.*, vol. xii, 1907, p. 80.

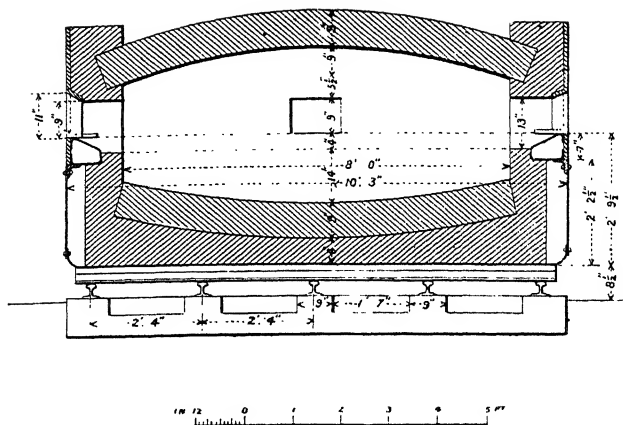


Fig. 224.—Transverse Section Port Pirie Softening Furnace.

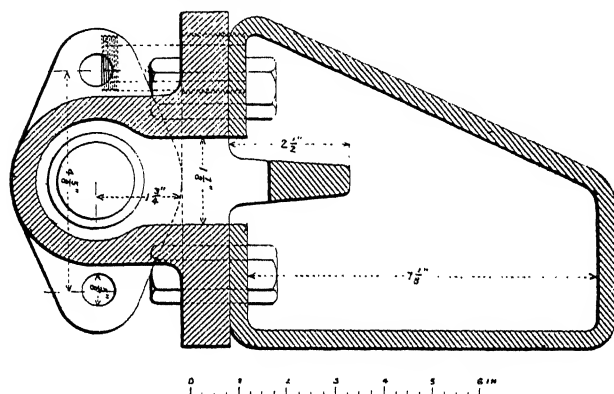


Fig. 225.—Detail of Water Jacket.

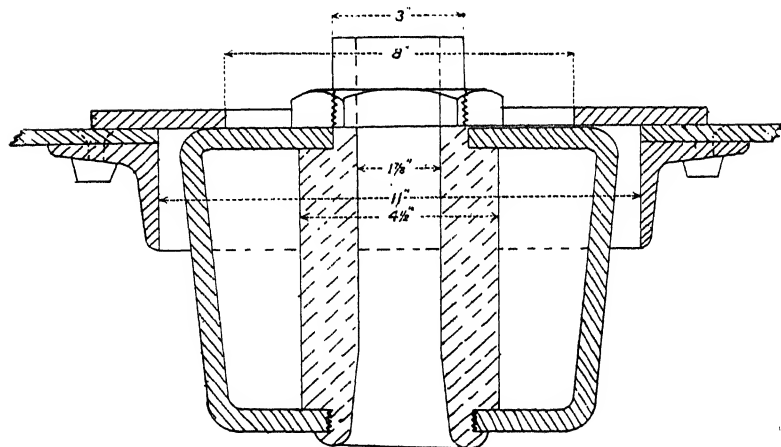


Fig. 226.—Detail of Tapping Jacket.

to resist the expansion. The furnace gases pass through the opening into a horizontal flue, which crosses the end of the furnace and leads into an independent downtake for each furnace connected with the main underground flue of the refinery. There are three doors on each side of the furnace which serve for charging and skimming respectively, and one skimming door in the end opposite the firebox. The hearth is built of the best Gartcraig firebrick as an inverted arch, and must be laid with the closest possible fireclay joints to make it lead tight; the working bottom is similarly built as shown in Fig. 224, and tightly wedged against its skewbacks in the side walls. Air is frequently admitted through the bridge and roof to accelerate oxidation, and sometimes also a blast is used under the grate when inferior slack is to be used as fuel.

The wrought-iron tank, inside which the furnace is built, rests upon a course of rails, as shown, and these upon a brick base, which in its turn is carried by a foundation of slag poured in from the blast furnaces, so as to form a solid block. Detail of the water jacket is shown in Fig. 225; it is made in two pieces, as shown, measuring about 7 inches high by 8 inches wide. The lower junction of the water jacket with the wall being a weak spot is re-inforced or protected by an overlapping apron of the outside cast-iron plates extending for 4 inches above the bottom of the jacket, the joint between being tamped with fireclay. In order to cool and preserve the roof, seven draught holes, 4 inches square, are left in the crown of the arch, the inflow of air through which can be controlled at will. Usually they are kept closed in the "drossing furnace," but open in the "antimony furnace," the temperature in which is higher.

American and English softening furnaces are very similar, except that in the latter a simple water pipe extending round the furnace inside the iron or steel tank is sometimes used instead of a jacket; and that in the former the water jacket is frequently a rectangular box, the inner side of which is formed simply by the side of the wrought-iron hearth tank, while the top and bottom are of square bar or of channel iron rivetted to it and to the outer plate. Figs. 227 to 230 represent an American softening furnace with rectangular water jacket, and end taphole passing through an extension of the wrought-iron jacket. Other details of the furnace are, in the main, similar to those of the Australian furnace previously figured.

The comparatively high temperature employed in the later stages of the softening and, still more, in the refining process (viz.,  $700^{\circ}$  to  $760^{\circ}$  C.) gives rise to a considerable corrosion of the hearth, and particularly of the sides of the furnace; the mixture of metallic oxides attacking the lining material to form silicates. In order to minimise this corrosion as far as possible, the densest aluminous firebricks are employed, set with the thinnest possible joints, since raw fireclay is always attacked much more readily than the brick. Recently the employment of *bauxite* brick for the roof and sides of softening and refining furnaces has become a common practice, with the result that the lining lasts from five to six times as long as ordinary firebrick, while the quantity of slag for re-smelting is much less, and, therefore, costs less for re-treatment. *Magnesia*, and especially *chrome magnesia*, bricks are also used for the portion of the lining inside the water jacket, extending for 2 or 3 inches above the level of the lead, and are found to resist very well the action of the highly corrosive antimony skimmings.

At *Peñarroya* (Spain), where furnaces of the non-water-jacketed pattern, like that shown in Figs. 220 and 221, are in use, holding 45 tons of lead, the lining of the sides and hearth of the furnace are of magnesia brick, the sides lasting four months and the bottom nine months; the roof is of aluminous semi-bauxite brick, and lasts much longer.

Instead of the water-cooled taphole, closed by a wooden plug, used in

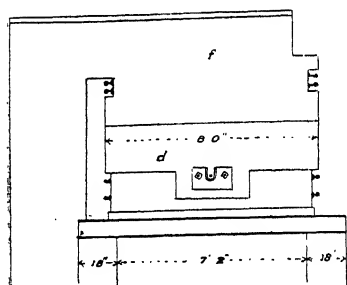


Fig. 227.

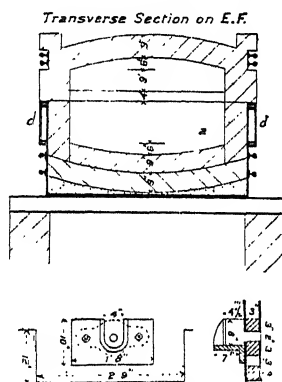


Fig. 228.

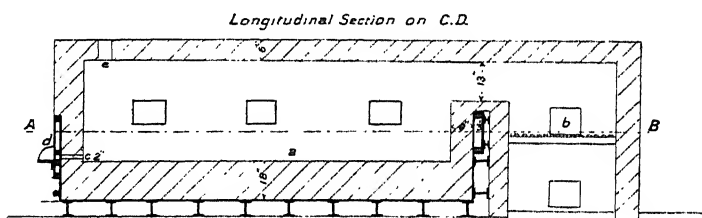


Fig. 229.

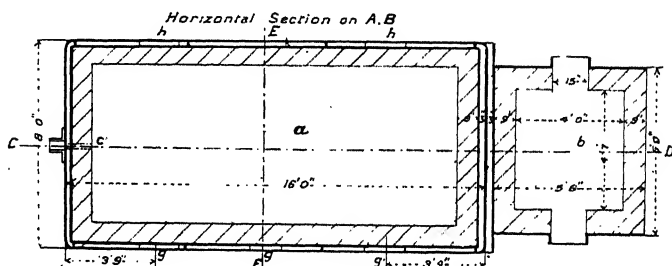


Fig. 230.

Figs. 227 to 230.—American Softening Furnace.

the Australian furnace figured, most softening furnaces have their tapholes arranged for an ordinary clay stopping, the premature break-out of which is guarded against by an iron plate sliding in a groove at the head of the launder that conducts the hot metal to the drossing and desilverising pots.



## METHOD OF WORKING.

In some few cases (as, for instance, at *Peñarroya*) the pig-lead is melted down in large drossing pots, the contents of which (after drossing) are tapped into the furnace for softening. It is, however, far more usual to melt down the pigs slowly in the furnace itself. The pigs of lead must be introduced carefully by means of a "peel" or long paddle, so as not to injure the working bottom, and within a period of from one and a-half to five hours after the charging is finished (according to the temperature employed) they are all melted and the *dross* ready for skimming. To assist in collecting the dross, a little fine coal is sometimes stirred in, and then the dross is all skimmed off by a rabble with long flat head resting in a hook suspended by a chain. The dross is of pasty consistency, and nearly black in colour owing to the presence of  $\text{CuO}$ ; it usually contains from 60 to 70 per cent. of lead when rich in copper (5 to 10 per cent.), but may run as high as 80 per cent. lead when the proportion of copper in the work-lead is low; its weight may vary from 3 to 6 per cent. of that of the charge; it is collected in slag pots or barrows, weighed, and wheeled to the liquation furnace. When the bullion has contained tin the dry *tin-arsenic* skimmings are next removed in the same way as the dross, the quantity of these, however, is usually so small as to be of no importance. The *antimony* skimmings are more fluid, looking like black oily drops upon the surface of the lead; the furnace-doors are, therefore, frequently opened to cool them, and permit of their more ready removal. Lime is sometimes thrown on to cool and thicken the mass, but this plan interferes with the subsequent liquation, and is, therefore, but seldom adopted. It is necessary to thoroughly stir the bath at frequent intervals, say at least every half-hour, for, otherwise, clean litharge would be shown, and the surface layer of lead become freed from antimony while that at the bottom was still very impure. Stirring and skimming must be repeated until the lead is found to be soft, as shown by the ladle test already described, which is often supplemented by taking out a portion of the skimming on a ladle or paddle, allowing to cool, and examining the fracture, which at first is black and glassy. A clean yellow colour with long, flaky crystals of litharge shows that the antimony is all "out," while a short-grained fracture with brownish colour or stain here and there indicates that a little more oxidation is required. The texture and appearance of the last skimmings, however, are not an invariably safe guide unless the whole be carefully examined, for a considerable portion of a skimming averaging 6 per cent. of antimony may show well developed flaky crystals of clean litharge, the antimony appearing to segregate in certain portions of a skimming.

To hasten the process, jets of steam have been sometimes introduced through perforated pipes. These have the ill effect of increasing the oxidation of lead, and, consequently, the amount of skimmings produced; a far more serious objection, however, is the increased corrosion of the furnace walls through the wave-motion set up in the bath, and of the roof through the projection of small drops of lead oxide. By far the most efficacious method of hastening the removal of antimony is to stir litharge into the bath at a comparatively low temperature, particularly the rich litharge from the finishing cupels and the second refining skimmings, which are very rich in metallic lead. This practice results in a considerable saving of time in

oxidation, besides affording a convenient means of disposing of these by-products. The "softening skimmings" will vary in amount according to the impurity of the bullion; usually, however, they do not amount to more than 5 to 8 per cent. of the weight of base bullion charged. The number of separate skimmings (*i.e.*, operations of skimming) to be made will vary entirely with the proportion of antimony in the lead; with comparatively pure lead only two may be required, but in bad cases as many as eight or nine may be run, keeping the charge in the furnace as long as 48 hours after the removal of the copper dross, and alternately heating up for oxidation and cooling for removal of the skimming.

The fuel usually employed is coal, the consumption of which varies with the degree of impurity of the work-lead and the size of the furnaces employed; in America, according to Hofman,\* it averages  $7\frac{1}{2}$  per cent. by weight on the base bullion treated. If this figure be correct, however, it should certainly not stand as a record of economical work, for the *Broken Hill Proprietary Company* at Port Pirie soften their impure bullion in twenty-four hours with a consumption of only 5 per cent. of coal; while at the *Howdon Works* (Newcastle-on-Tyne) bullion with 3 per cent. impurities is softened in 140-ton furnaces with a coal consumption of only  $2\frac{1}{2}$  per cent. by weight.† Crude petroleum or residue oil sprayed by a Körtzing injector, or, better, by a jet of compressed air is used in some American works with good results, the consumption of oil being, according to Hofman,‡ about 2·7 per cent. by weight.

The average time taken to soften ordinary American lead bullion is fourteen hours. In the United States the usual charge is 30 to 40 tons (rarely 50 tons), and the lead softened thus quickly at a comparatively high temperature is run direct from the furnace into the zincage pots, an arrangement found also at some Continental and English works. Hutchings recommends § the use of much larger furnaces taking charges up to 140 tons and softening slowly at a lower temperature. By this method, even with work-lead containing 3 per cent. of impurities, taking forty-eight hours to soften, the fuel consumption is only about  $2\frac{1}{2}$  per cent. by weight, as compared with the American consumption of about  $7\frac{1}{2}$  per cent.; the saving in fuel being much more than the loss in interest. Hutchings further recommends casting the lead from the softening furnace and remelting it in the desilverising kettles as affording opportunity for an additional skimming of coppery dross, yielding a purer lead and saving zinc. The same object may, however, be attained by what is called "ringing" the lead, or allowing it to cool gradually in the desilverising pot almost to the solidification point, and removing the cooled ring for remelting and drossing before heating up the remainder for the first zining; or, as pointed out by Huntington,|| by interposing a collecting pot between the softening furnace and desilverising pot in which this cooling and drossing takes place, bringing down the copper contents of the bullion to 0·08 per cent., even when originally containing as high as 0·45 per cent. The latter writer further points out that the 140-ton furnaces would be out of place in medium-sized works where the softening furnace forms only a part of one continuously working whole; though suitable enough

\* *Metallurgy of Lead*, 1906, p. 445.

† Hutchings, *E. and M. J.*, Feb. 3, 1896.

§ *E. and M. J.*, Feb. 23, 1895.

‡ *Op. cit.*, 1906, p. 445.

|| *E. and M. J.*, March 30, 1895.

for a desilverising plant, pure and simple, working on a sufficiently large scale.

At the *Port Pirie* refinery of the *Broken Hill Proprietary Company*,\* the average bullion treated formerly contained 0·8 per cent. Cu, 0·7 per cent. Sb, 300 ozs. Ag, and 5 dwts. gold per ton (*v. analysis*, No. 7, Table XXIV.); now that ores from the deeper levels are worked it is much purer, averaging only 0·45 per cent. Cu, 0·2 per cent. Sb (*analysis*, No. 8, Table XXIV.). At these works two separate sets of furnaces are employed for drossing and softening respectively, the latter being built on a slightly lower level, about 1 foot below the former. There are two drossing furnaces arranged lengthwise on the uppermost level, each of which supplies a pair of softening furnaces of the same dimensions placed end on. All the furnaces are of the same type, and approximately of the same dimensions; they have been illustrated in Figs. 222 to 226.

The charge for a drossing or "copper" furnace is 36 tons (*i.e.*, of 2,240 lbs. = 40 tons American), which takes about three hours to charge in carefully with long paddles, so as not to injure the bottom. As the furnace is kept at a very low and uniform temperature, the charge takes about four hours to melt down; as soon as this operation is completed the doors are thrown open so as to prevent it from getting any hotter, and the dross, which (after liquation) weighs about  $1\frac{1}{2}$  tons, is skimmed off. The composition of this dross is given in Table XL., and its treatment will be referred to later. It is skimmed into old slag pots with a taphole in the bottom, through which the lead which drains from it can be tapped into moulds and returned to the furnace, by means of a skimmer which resembles a small perforated inverted rectangular tray on a long handle. The consumption of coal in the copper drossing furnace is 14 cwts. for a charge of 36 tons, or only 2 per cent. The "copper" furnace is provided with two water-jacketed tapholes on the long side facing the softening or "antimony" furnaces, and as soon as one of these is ready for a new charge the lead is run into it by means of a movable clay-lined iron gutter from the nearest taphole of the drossing furnace, any litharge or skimmings which it is intended to add as oxidising agents for the antimony being charged at once.

The drossed bullion sent to the softening furnace now contains Cu 0·125 per cent., As 0·094 per cent., Sb 0·18 per cent., Ag 70 to 80 ozs. per ton, and Au 1 dwt. per ton. The drossing furnace being kept at a fairly even temperature throughout, the wear and tear on it would be quite slight but for the formation of coppery accretions which form at about the level of the surface of the bath, and have to be chipped off from time to time, for which purpose the furnace has to be cooled down; the chippings go to the refinery blast furnace.

The process of softening in the antimony furnace is carried on in the usual manner, with alternate hard firings for oxidation and coolings for skimming. Formerly, when the lead contained 0·7 per cent. of antimony, five skimmings were made, and the process took 18 hours; now, as a rule, only two to three skimmings are taken off. The first firing usually lasts about six hours, after which the bath is poled and thoroughly stirred by means of iron rakes which are plunged down the sides and up the centre. The doors are then thrown

\* *Private Notes*, 1896; also Bayly, *Trans. Aust. M.E.*, vol. xii., 1907, pp. 80, *et seq.*, and Poole, *Queensland Govt. Mining Journal*, March 15, 1909, p. 117.

open to cool the bath, and the first skimming is taken off. The fire is then cleaned, and another four hours' hard firing given, after which the cooling is repeated and a second skimming is taken off; occasionally only the portion of this skimming near the door will be dark and antimonial, the rest being clean and free from antimony; more frequently, however, the second skimming is all antimonial, and a third small skimming has to be taken off to clean the lead. The bath is then allowed to cool off or "go back" before being tapped into the zincing kettles, its composition being Cu 0.10 per cent., As nil, Sb 0.089 per cent.

Owing to the high temperature and the corrosive effect of the litharge the wear and tear on the antimony furnace is heavy, the average life of the lining bricks at the level of the litharge being only four weeks. The cutting of the brick is rapid until the nose of the water jacket is just exposed, after which it is much slower, but it does not do to carry the level of the bath much above the level of the nose of the jacket, since the cooling effect then becomes too great. The coal consumption is about 22 cwts. per charge of 36 tons, or just 3 per cent., making a total of 5 per cent. by weight for combined drossing and softening. Each furnace is worked by one man per shift. The loss by volatilisation, working at this low temperature, is very slight, and the system of using separate drossing and softening furnaces has much in its favour besides economy of fuel, for the drossing furnace, if carefully handled, lasts indefinitely, and even on the softening furnaces repairs are much lighter, owing to the almost constant temperature at which they are kept. The complete separation of copper in the first furnace, moreover, much simplifies the subsequent treatment of bye-products, particularly of the antimony skimmings. Furthermore, the perfection of the softening, coupled with the low temperature in the desilverisation pots, renders it possible to produce very rich gold and silver crusts, which means less lead to be cupelled and smaller losses of silver, less dross is also produced in the retort.

A somewhat similar method is in use at *Hoboken* (Antwerp)\* for treating a base antimonial and arsenical coppery bullion containing 5 to 6 per cent. Sb, 2 to 3 per cent. As, and 0.5 to 1.0 per cent. Cu. Before softening, it is liquated in a separate liquating furnace, where it yields a dross of the composition given in Table XL. The liquated bullion then goes to the ordinary softening furnace to remove antimony.

At *Peñarroya* (Spain)† the furnace work-lead is melted down slowly in 50-ton drossing kettles, where a coppery dross is removed, and is then brought to the softening furnace in 25-ton ladles handled on trucks by a locomotive, and from the trucks to the furnaces by means of a fixed lift. Before the charge of 45 tons lead is run into the softener from the ladles, about half a ton of zincy powdery litharge from the refining furnace is put into the furnace, which much assists in the expulsion of antimony. Softening takes twelve hours, and yields from 1 to 2 tons of skimmings.

The composition of various samples of dross and of antimony skimmings is given in Table XL.

When treating work-lead rich in gold, such as is produced regularly at certain Australian works and occasionally at some American plants, the dross from the softening furnaces is also quite rich in that metal, and richer

\* Firket, *Ann. des Mines de Belgique*, 1901, p. 237.

† *Private Notes*, 1908.

the higher the proportion of copper in it. Thus when treating work-lead carrying an average of 12 ozs. of gold per ton, and producing drosses with an average lead content of about 67 per cent., if the copper present in the dross is only about 1 per cent., the gold assay of the dross will not average much above 2 ozs. per ton, whereas when the dross contains 4 or 5 per cent. of copper its assay in gold will run up to 5 or 6 ozs. per ton. In order to avoid locking up precious metals in this product and throwing them back to be exposed to the losses of blast-furnace treatment, at some smelting works, according to Hofman,\* galena is added to the softening furnace with the object of transforming the dross into a matte which is much lower in silver and particularly in gold. This modification, however, does not appear to be particularly good practice, inasmuch as the temperature of the softening furnace during the drossing operation, if this be conducted so as to give the best results, will not be high enough to form a matte, and it would seem better to treat the dross separately with galena, as described on a subsequent page, returning the bullion thus reduced to the softening furnace.

TABLE XL.—ANALYSES OF COPPER DROSS AND ANTIMONY SKIMMINGS.

Reference,	COPPER DROSS.				ANTIMONY SKIMMINGS.			
	Globe, Denver.		Port Pirie, Liquated.	Hoboken, Antwerp.	Globe, Denver.	Port Pirie.	Hoboken, Antwerp.	
	Before Liqua- tion.	Liquated						
	1	2	3	4	5	6	7	8
Pb, . . .	53·0	50·0	77·5	67·70	..	74·0	50·65	72·50
Ag, . . .	..	0·257	..	0·18-0·36	..	..	0·09-0·27	..
Cu, . . .	18·2	26·8	8·0	14·16	4·1	..	..	0·29
Sb, . . .	..	0·18	0·7	2·4	17·0	10·7	15·28	6·91
As, . . .	..	7·31	0·7	5·8	1·1	1·0	0·1-0·5	1·65
Zn, . . .	..	..	1·2	..	0·9	..	..	..
S, . . .	2·0	3·6	4·3	..	5·7	..	..	..
Fe, . . .	1·2	..	1·0	0·5-2	4·3	..	..	0·29
Al <sub>2</sub> O <sub>3</sub> , . .	..	..	..	..	..	..	..	0·87
CaO, . . .	..	..	..	..	..	..	..	1·10
Insoluble, .	0·6	4·8	2·1	4·8-7·2	3·8	..	..	4·10

*References.*—1 and 5. Iles, *Metallurgy of Lead*. 2. *Private Notes*, 1897. 3 and 6. Delprat, *Trans. Aust. I.M.E.*, vol. xii., 1907, pp. 1-27. 4 and 7. Firket, *Annales des Mines de Belgique*, 1901, pp. 237, et seq. 8. Piddington, *Journ. Ch. and Met. Soc. S. Africa*, 1903; iii., pp. 207-210.

**Refining in Kettles.**—Refining at Lautenthal has been already described. Whenever the work in the softening furnace has been very thoroughly done, the lead, after desilverisation, may be economically refined in pots, as it is antimony and not zinc which brings about the greatest destruction of pots.

At the *Port Pirie* refinery, kettle refining was until recently practised.† The kettles (here called “pans”) were 8 feet 6 inches diameter and 3 feet

\* *Op. cit.*, 1906, p. 435.

† *Private Notes*, 1896; see also Delprat, *E. and M. J.*, March 16, 1907, p. 518.

4½ inches deep, holding, when full, 30 tons each; they were provided with movable steam pipes reaching to the bottom, and with a hood similar to those at Lautenthal. The life of each pot was about thirty-five charges, or about 850 tons of lead refined, the thickness of cast iron being 1½ inches at the bottom and 1¼ inches at the rim, and the weight, when new, 52 cwts. each. Steel kettles have been tried, but were not a success—possibly on account of poor quality of steel.

The usual charge for one of these kettles was about 24½ tons of desilverised lead from the zincage pots, containing 0·67 per cent. Zn, which, by strong firing, was brought to a dull red heat; the hood was then lowered and steam blown through for four hours, at the end of which time the lead had become comparatively cool; the hood was then raised so as to remove the first “refining skimmings,” which amounted generally to about 2,400 lbs. (or, say, 5 per cent. of the weight of lead charged) and contained 75 per cent. lead and 15 per cent. zinc. The lead was then again heated red hot and steam again passed for half an hour. The second skimmings then removed weighed about 350 lbs. and contained only 0·5 per cent. zinc. The lead was then maintained at an even temperature and cast when convenient. As a rule, each charge took about sixteen hours to work off (including casting) and consumed about 12 cwts. of coal, or 2½ per cent. of the weight of lead charged, (according to Hofman,\* 6 per cent. is required in American works).

At the *Pueblo Smelting Works* (Pueblo, Colo.) similar kettles to those at Port Pirie are in use, but somewhat smaller—viz., 8 feet 1 inch diameter by 3 feet deep, and holding 20 tons apiece; the flanges have two concentric raised rings, between which the hood comes down to make a joint. The lead is raised to a cherry-red heat, which takes three hours, and steam is then turned on for two hours, producing about 1,700 lbs. of oxide skimmings, which are removed by the Howard skimmer.†

On account of the heavy wear and tear of kettles already referred to in discussing the practice at Lautenthal, and inseparable from the temperatures of 700° to 760° employed, the method has been given up almost everywhere, for instance at *Monteponi* (Sardinia) and *Peñarroya* (Spain), at both of which places furnaces are now employed similar in all respects to those used for softening.

**Refining in Reverberatories.**—The same furnace is used for softening, but it is frequently made 3 inches shallower. The desilverised lead is syphoned into it by means of the Steitz lead syphon made of gas pipe, one form of which is shown in Figs. 230 to 231, or by means of the Rösing lead pump, if the works are on a level. The mode of operating is similar to that in the softening furnace, except that a higher temperature is necessary, and that zinc, and not antimony, is the principal metal to be oxidised. The last traces of antimony, however, when the lead has not been thoroughly softened, are very difficult to remove in refining, and, therefore, a lead containing 0·6 per cent. of zinc and 0·3 per cent. of antimony will yield as much as 4 or 5 per cent. of skimmings containing about 90 per cent. of lead. These skimmings are removed in separate “heatings,” the doors being thrown open for a time to cool the skimmings and render them easier to remove. Three heatings are usually required, taking about twelve hours, after which the litharge drawn from the surface of the bath should be bright yellow and

\* *Op. cit.*, 1906, p. 468.

† *Mineral Industry*, ii., p. 440.

free from brown spots (antimony). Steam is now rarely used in refining furnaces, for the same reasons as have been given in connection with softening furnaces. The wear and tear on refining furnaces is much heavier than on softening furnaces, in consequence of the higher temperature necessarily employed, and much trouble is caused by the formation of zinc accretions at the level of the bath, which have to be chipped out at somewhat frequent intervals.

The furnace employed at *Peñarroya* \* is of the pattern shown in Figs. 220 and 221, it holds 40 tons of desilverised lead, and has no water jacket, but the hearth and sides are built of magnesia brick, which lasts from  $2\frac{1}{2}$  to 3 months on the sides, the roof being of aluminous semi-bauxite brick, which lasts 10 to 12 months, as also does the magnesia hearth. Refining and casting here take only twelve hours together.

At the *National* † plant (S. Chicago) reverberatory furnaces of two sizes are used for refining, the charge being either 28 tons (finished in four to five hours) or 80 to 90 tons (finished in eight to ten hours). Steam is introduced through  $\frac{1}{2}$ -inch iron pipes, of which there are four in the small and eight in the large furnaces. The first skimmings contain much antimony, and are smelted together with the softening skimmings, the zincy litharge which follows is reduced in a reverberatory to a second quality lead.

The present practice at *Port Pirie* ‡ is to refine in a furnace exactly like the softening furnace, and the refining, like the softening, is conducted by means of alternate heavy firing to oxidise, followed by cooling to take off skimmings, here called "lead dross." Two skimmings are usually made, after the first and before the second the lead bath is tested for softness by stirring up, ladling out a sample, skimming with a board, pouring into a taper mould  $8 \times 2 \times 1$  inches, and again skimming. Antimony is shown by a frosted white spot in the centre of the cooled surface, zinc by fern-like crystals on the surface, and by the absence of the characteristic iridescent blue tarnish which is only obtainable with pure lead. The first skimming is usually removed after about five hours' hard firing and half-an-hour's cooling, after which the furnace is again fired hard. The total time required for the refining process is from 12 to 16 hours, and the coal consumption is 4 per cent. The refining dross contains Pb 85 per cent., Zn 11 per cent., Ag 0.1 to 0.2 oz., and is sent back to the refinery blast furnace.

The average composition of the market lead produced at Port Pirie during one whole year was as follows :—Cu 0.00026 per cent., Ag 0.00096 per cent., Zn 0.00187 per cent., Sb 0.00325 per cent., Pb (by difference) 99.99366 per cent.

**Casting.**—The operation of casting (or "moulding" as it is often termed) from refining kettles is a simple one, the syphon employed being more or less similar to that shown in Fig. 231, the flow of which can be slackened or stopped entirely by raising the end of the movable arm. In use, this syphon must be firmly clamped to the kettle by means of two hoops which encircle it.

The form of syphon figured is, however, old-fashioned; the more modern form has a stopcock on each arm, and the outer end, instead of having upon

\* *Private Notes*, 1908.

† Pufahl, *E. and M. J.*, April 14, 1906, p. 718.

‡ Bayly, *Trans. Aust. M.E.*, vol. xii., 1907, p. 90; and Poole, *Queensland Govt. Mining Journal*, March 15, 1909, p. 118.

it a loose joint with delivery pipe attached, delivers into a cup upon the end of a long independent horizontal pipe which swings radially and can, there-

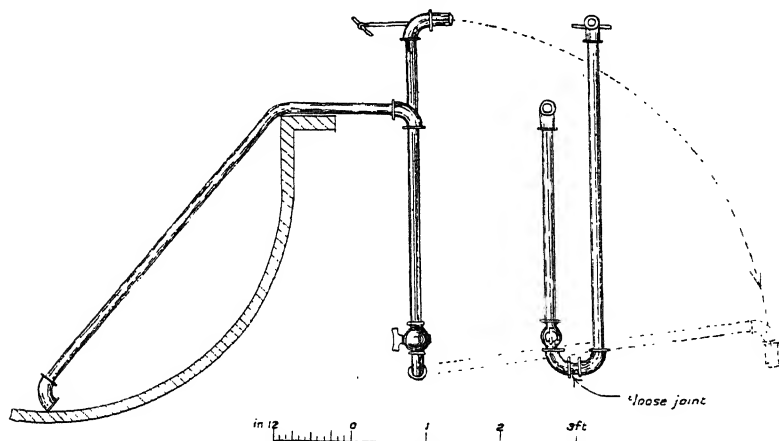


Fig. 231.—Lead Syphon.

fore, deliver into a semi-circle of moulds while the flow from the syphon remains uninterrupted. The stopcocks on the syphon are for filling, which can only be done after the syphon itself has become thoroughly heated by lying upon the molten lead in the kettle for some time before it is required.

When the lead has been refined in a reverberatory furnace, it may be cast direct from the furnace. The apparatus used for this purpose is shown in Fig. 232. This method of casting has obvious advantages, but it has the grave disadvantage of requiring the refining furnace to be cooled down for moulding with a loss of, at least, the two hours required to heat it up again, besides the increased strain on the furnace and wear and tear of the lining. Hence it is decidedly preferable to tap into a warmed market-pot and cast at leisure when the lead has attained the right temperature, either by means of the Steitz syphon or of the Rösing

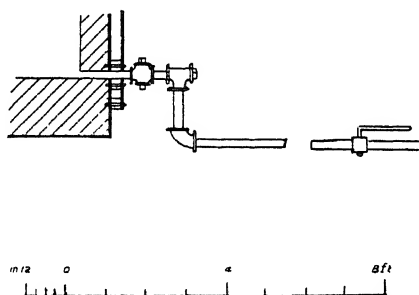
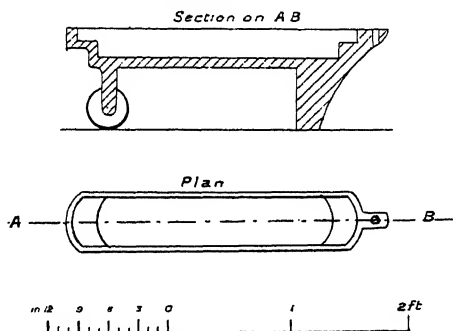


Fig. 232.—Apparatus used in direct casting.



Figs. 233 and 234.—Mould.



lead pump.\* The moulds used are arranged 40 to 60 at a time in a great semi-circle round the end of the radially-moving end of whatever casting apparatus is employed. Sometimes they are mounted on wheels, like those shown in Figs. 233 and 234; nowadays, however, the ordinary form of mould, shown in Fig. 195, is often employed, arranged upon a semi-circular frame, so as to be readily inverted for delivery of the pigs upon the floor.

At *Port Pirie* the merchant kettle, into which the lead from the refining furnace is tapped, is of 35 tons capacity. The usual type of syphon is used, made of  $1\frac{1}{2}$ -inch pipe,  $\frac{1}{4}$  inch thick, and two sizes of bars are made, one being the usual 100-lb. size for the English market, for which the lead lowest in silver is selected, and a 200-lb. size for the Chinese market, for which is used lead containing up to  $\frac{1}{2}$  oz. silver, since such small silver contents are considered in the price by the Chinese purchaser.

Some data with reference to softening furnaces at different works are given in Table XLI.

TABLE XLI.—WORK DONE IN SOFTENING AND REFINING FURNACES.

	Freiberg, Saxony.	Omaha and Grant, Omaha, Neb.	Port Pirie, S.A.	Howdon Works, Newcastle -on-Tyne.
Reference, . . . .	1	2	3	4
<i>Softening—</i>				
Weight of charge in furnace, . . tons,	21	45	36	140
„ dross removed, . . „	..	..	$1\frac{1}{2}$	..
„ skimmings produced, . . cwts,	..	..	36	..
Number of hours in furnace, . . .	36	12	27	48
Percentage of softened lead, . . .	85	..	86	..
Weight of coal used per charge, . . tons,	..	$1\frac{1}{2}$	$1\frac{3}{4}$	..
Percentage consumption of coal, . . .	..	3.3	5.2	$2\frac{1}{2}$
Men employed per 24 hours, . . .	..	3	3	..
Percentage of antimony in skimmings, .	..	..	10	..
<i>Refining—</i>				
Weight of charge, . . . . tons,	..	..	30	..
„ skimmings removed, . . cwts,	..	..	24	..
Percentage of market lead, . . .	..	..	95	..
Time taken in refining, . . . hours,	9	12-18	16	..
Percentage of Zn before refining, . . .	0.75	..	0.70	..
„ „ after . . . .	0.0002	..	0.00015	..
Silver in market lead, . . . per cent.,	0.0007	..	0.0009	..
„ „ „ dwts. per ton, . . .	$4\frac{1}{2}$	..	$5\frac{1}{2}$	..
Percentage consumption of coal, . . .	..	6	$3\frac{1}{2}$	..

References.—1. Plattner, *Oesterr. Z. f. B. u. H. W.*, 1887, p. 421. 2. *Private communication*. 3. *Private Notes*, 1896. 4. *E. and M. J.*, Feb. 23, 1895.

**Bye-products of the Softening Furnace.**—The ordinary bye-products of the softening furnace are two—viz., copper dross and antimony

\* This appliance, but rarely seen in recently laid-out works with proper fall, is to be found at many old European works originally laid out upon a level for hand work only, see *Berg- u. Hüttenmannische Zeitung*, 1889, p. 262.

skimmings; sometimes, as at Freiberg, a third intermediate product of "tin skimmings," but usually traces of tin go into the ordinary antimony skimmings.

**Copper Dross.**—This is usually worked up by liquating at a low temperature in a liquating furnace, which is frequently built like a small softening furnace, but kept only for treating the bye-products. After as much lead has melted out as possible, the dry dross floating on the metallic bath is raked off, and a fresh charge of dross added; the lead is tapped out into a pot from time to time, and goes back to the bullion-softening furnace.

The liquated dross varies in composition according to the nature of the bullion from which it has been derived; some analyses of this substance have been given in Table XXXIX. It usually contains about two-thirds the silver contents of the unsoftened bullion, and, perhaps, 80 per cent. of lead, with 10 or 12 per cent. of copper. The way in which gold becomes concentrated in copper dross, owing to the strong affinity of copper for gold is remarkable, the dross frequently assaying higher in gold than the bullion from which it is derived.

It is usually smelted in the refinery cupola to copper matte, together with sulphide ores or, preferably, as at *Port Pirie*, with low grade ore-matte, which it raises in copper contents, at the same time that a comparatively clean work-lead is produced carrying most of the silver and gold in the charge.

At *Peñarroya* the dross from the softening furnaces after liquation and the retort drosses from the Parkes process are smelted in reverberatory furnaces enclosed in iron pans, together with sufficient galena to form a matte which contains all the copper, the lead reduced, carrying practically all the silver, being returned to the softening furnace.

At the *National* plant (S. Chicago)\* the copper dross is smelted in reverberatory furnaces, together with 80 per cent. by weight of galena for work-lead and 35 per cent. copper matte. The latter is enriched to 55 per cent. by resmelting with roasted matte, and is then smelted in a reverberatory concentrating furnace to 95 per cent. copper, which is cast into plates and used in the parting process.

**Antimony Skimmings.**—These are also first liquated, with the double object of extracting the shots of metallic lead mechanically entangled, and of desilverising the skimmings, which, before liquation, contain 6 to 15 ozs. silver per ton with base bullion of 300 ozs. Frequently the same furnace is employed as that used for liquating dross, the flame being made as smoky as possible in melting down, in order to reduce lead and carry down the silver; a little fine coal is also thrown on the surface of the bath with the same object. The contents of the furnace are tapped into a large overflow pot with a spout, in which the lead collects, while the liquated skimmings overflow into slag pots. An antimonial lead-copper speiss is often found beneath the cones of cooled skimmings, which is worked up by smelting with the dross. When the bullion softened is exceptionally rich in gold (say 10 to 15 ozs.) the antimony skimmings will also be comparatively rich in that metal, and carry sometimes from 1 to 2 ozs. per ton. This can be, however, pretty thoroughly eliminated and carried down in the lead reduced by using more coal than usual in the liquating or smelting furnace, as subsequently described.

\* O. Pufahl, *Oest. Zeits. f. B. u. H. W.*, 1905, p. 400.

The liquated skimmings, now containing only 2 to 3 ozs. of silver per ton, are smelted with a calcareous slag from a previous campaign in a small water-jacketed cupola, together with sufficient galena to reduce all the antimony and lead in the skimmings to metal without forming matte. The slags produced being fusible, about 11 per cent. only of fuel is usually required on the charge, but generally as much old slag is used as new skimmings. Too ferruginous a slag and too high a reduction is liable to give rise to the formation of an antimonial speiss containing up to 30 per cent. Sb. The resulting products are hard lead or antimonial metal, containing from 7 to 25 or more per cent. of antimony, and an antimonial leady slag, which is saved for the next campaign.

The crude hard lead usually contains a good deal of copper, and is best melted down at a low temperature and drossed to get out as much copper as possible; after which it is poled in the tap kettle to remove the remainder, and sold to typefounders or used for making alloys like Babbitt and Britannia metals.

Instead of merely liquating the antimony skimmings with very little reduction of combined lead, it is more common nowadays to smelt them in a reverberatory furnace, built like a softening furnace, but never tapped quite dry, so that successive charges are dropped upon a lead bath and melted down rapidly. The charge consists of raw skimmings, galena, and fine coke or coal sufficient in amount to reduce enough lead to thoroughly free the refined skimmings from both copper and silver, as well as gold, if present. This method is followed at *Peñarroya*,\* *Hoboken*, and other places on the Continent.

At the *National* plant, S. Chicago,† antimony skimmings are smelted in a reverberatory furnace with coke breeze, producing work-lead and antimonial slag, which is reduced to hard lead containing 27 per cent. antimony and 0.5 per cent. each of arsenic and copper in a small cupola with eight tuyeres. The percentage of copper in this lead could no doubt be reduced by making matte in the reverberatory furnace.

In the United States of America the commonest way of treating dross and skimmings from the softening furnace is to smelt both together in a reverberatory furnace with a small quantity of galena ore low in silver. The products are work-lead, containing most of the silver and gold contents of both dross and skimmings; high grade copper-lead matte, free from iron; and refined antimony skimmings, comparatively free from both silver and copper,‡ and, therefore, suitable at once for smelting to antimonial metal. The furnace has two tapholes, from the upper of which the refined skimmings are tapped, the lower one serving for lead and matte. The lead goes to the softening furnace and is treated like the ordinary work-lead.

The copper-lead matte is charged into an English cupelling furnace (*v. Chap. xvii.*) and oxidised, with the addition of siliceous sand or broken quartz. The products are lead slag, which runs off continuously into slag pots, and a mixture of 60 per cent. copper matte with some metallic bottoms, which collects on the test. When full, the contents of the test are tapped into moulds; the copper bottoms, containing all the gold of the dross and

\* *Private Notes*, 1908.

† O. Pufahl, *Oest. Zeits. f. B. u. H. W.*, 1905, p. 400.

‡ Although, however, rarely so free from copper as when the dross and skimmings are smelted separately.

skimmings, are worked up by solution processes; \* while the gold-free copper matte, rich in silver, is cupelled up to argentiferous metallic copper in another hearth, and used as a precipitant for the silver in parting doré silver † bullion. In this way all the metallic constituents of these by-products are utilised in the most advantageous way, except the part volatilised, which is rather more than usual, owing to the high temperature and intense oxidation of the cupelling hearth.

At *Port Pirie* (S.A.) ‡ the antimony skimmings are smelted in a small reverberatory furnace built in an iron tank like a softening furnace, but without a water-jacket, and measuring 11 feet in length by 6 feet 6 inches wide, the nominal capacity being 10 tons. The furnace is provided with two tapholes, and twelve charges are run through in twenty-four hours, each being composed of 2,800 lbs. skimmings and from 110 to 120 lbs. each (generally 115 lbs.) of fine coal and of coke breeze; the lead from four successive charges is allowed to accumulate in the furnace, and is only tapped at every eight-hour shift, while the slag accumulates till, at the end of twenty-four hours, the whole charge is tapped out into a tap pot. The product from each ton of skimmings is 1,200 lbs. of liquated lead containing on an average Cu 0.12 per cent. (formerly  $1\frac{1}{2}$  per cent.), Sb 0.66 per cent. (formerly 1 to 2 per cent.), Zn 0.0046 per cent., and all the silver in the skimmings, which, however, does not amount to more than about 10 ozs. per ton. The remaining lead contents of the original skimmings (except about 7 or 8 per cent. loss by volatilisation) are contained in the "antimonial slag" (elsewhere called "liquated" or "refined" antimony skimmings) which contains 46 to 56 per cent. lead and 17 to 27 per cent. antimony, besides 3 to 7 per cent. As, 0.7 per cent.  $\text{SiO}_2$ , and Ag about 2 ozs. per ton.

The "antimonial slag" is then smelted with fluxes in an 84 by 36-inch 10-tuyere furnace, run like an ordinary lead furnace. The regular charge is 450 lbs. of antimonial slag, 250 lbs. returned slag from a previous operation, 100 lbs. ironstone, 50 lbs. limestone, and 10 to 15 lbs. scrap iron, which is smelted with 90 to 100 lbs. of coke, blast being used at only 3 to  $4\frac{1}{2}$  ozs. pressure. These charges are run at the rate of one every fifteen minutes, or 32 per 8-hour shift, equals 63.6 short tons per twenty-four hours, producing 70 to 75 bars of antimonial metal per shift, containing from 20 to 24 per cent. antimony and  $1\frac{1}{2}$  per cent. arsenic. Analyses of typical samples of the antimonial metal and of the slag are as follows:—

<i>Antimonial Metal.</i>		<i>Slag.</i>	
Pb, . . . .	76.04 per cent.	Pb + Sb, . . . .	6.21 per cent.
As, . . . .	1.84 "	$\text{SiO}_2$ , . . . .	30.5 "
Zn, . . . .	0.13 "	$\text{FeO}$ , . . . .	26.92 "
Cu, . . . .	0.26 "	$\text{MnO}$ , . . . .	3.80 "
Sb, . . . .	21.42 "	$\text{CaO}$ , . . . .	9.50 "
Ag, . . . .	1 oz. per ton.	$\text{Al}_2\text{O}_3$ , . . . .	17.58 "
		Zn, . . . .	4.97 "

The antimonial metal is run into a drossing kettle 4 feet diameter, and melted down at a low temperature. A ring or sleeve of iron 3 feet diameter, 18 inches deep and 1 inch thick, is then floated on top, and the metal inside

\* For a description of which the volume on the *Metallurgy of Silver* should be consulted.

† i.e., silver containing gold.

‡ *Private Notes*, 1896; also Bayly, *Trans. Aust. I.M.E.*, vol. xii., 1907, pp. 80, *et seq.*

carefully skimmed; lead from the furnace lead well is then tapped into the pot outside the ring, where the dross rises, whilst, after carefully skimming the surface, clean metal is dipped from inside the ring and poured into moulds, where it is again skimmed, the bars then being sold to typefounders.

The composition of some hard leads and hard lead drosses is given in Table XLII.

The treatment of tin skimmings at Freiberg (which contain PbO 70·35, SnO<sub>2</sub> 12·53, Sb<sub>2</sub>O<sub>3</sub> 12·50, As<sub>2</sub>O<sub>3</sub> 4·73, CuO 0·61 per cent., and Ag 70 ozs. per ton) consists of a succession of softenings in a reverberatory and smeltings of the skimmings in a cupola, which need not be described here. The final product is a desilverised alloy, containing Sn 33 per cent., Sb 14 per cent., As 1 per cent., which is readily marketable. For details the student may consult the original paper of Plattner.\*

TABLE XLII.—ANALYSES OF HARD LEADS AND HARD LEAD DROSSES.

	Freiberg.	Lautenthal.	Przibram.	Broken Hill.	Pueblo S. & R. Co., Pueblo.	La Tortilla, Linares.	Clausthal Dross.
Reference,	1	2	3	4	5	6	7
Pb, . . . . .	81·5	82·08	74·886	78·0	74·886	85·5	22·962
Ag, . . . . .	..	0·003	0·009	0·07	0·008	0·001	..
Sb, . . . . .	15·0	17·34	24·270	20 to 26	20·550	11·84	38·763
As, . . . . .	2·5	..	0·109	1 to 2	0·109	0·89	..
Cu, . . . . .	trace	0·62	0·160	trace	0·300	1·55	37·643
Ni, . . . . .	..	..	0·015	..	..	0·12	..
Fe, . . . . .	..	..	0·018	trace	..	..	0·139
Zn, . . . . .	..	..	0·009	..	..	..	0·232
Sn, . . . . .	0·3	..	0·524	..	..	..	..
S, . . . . .	..	..	..	..	..	..	0·240
Au, ozs. per ton, .	..	..	..	..	..	2 dwts.	..
	99·3	100·043	100·000	..	95·853	99·901	99·979

References.—1. J. f. d. B. u. H. W. in Sachsen, 1883. 2. Schnabel, *Handbuch der Metallurgie*, vol. i, p. 434. 3. *Oesterr. Jahrb.*, vol. xxxix, p. 64. 4. *Private Notes*. 5. Hofman, *Metallurgy of Lead*, 1906, p. 498. 6. *Private Notes*, 1908. 7. B. u. H. Z., 1870, p. 169.

**Bye-products of the Refining Furnace (or Kettle).**—In refining there is only one product—namely, *refining skimmings*—which is, properly speaking, a bye-product of the Parkes process, but may be conveniently referred to here.

These skimmings are mixed with 10 per cent. of fine coal and smelted, preferably in an English reverberatory furnace. The temperature is gradually raised till no more lead runs, when the residues are rabbled, collected, and drawn. The lead goes back to the refining furnace, the residues, which, besides lead, contain much zinc and some antimony, as well as traces of

\* *Berg. u. Hüttenm. Zeitung*, 1883, p. 417; also abstract of the same in Schnabel, *Handbuch der Metallhüttenkunde*, vol. i, p. 431.

other metals, are treated together with the softening skimmings, or returned to the ore blast furnace together with litharges, &c., as lead flux for siliceous ores.

At *Port Pirie* the first and second refining skimmings are treated separately. The former, containing 75 per cent. Pb, 15 per cent. Zn, and about 0.3 oz. silver per ton, are treated in the refinery blast furnace, together with litharges, mattes, and other products. The latter, containing 96 per cent. Pb, 0.5 per cent. Zn, and  $\frac{1}{2}$  oz. Ag per ton, are added to the softening furnace, where they assist in the expulsion of antimony. This would seem to be about the best possible way of handling these second skimmings. In custom smelting works treating chiefly ores of the precious metals, when lead is comparatively scarce, refining skimmings are frequently sent back directly to the charge of the ore-smelting blast furnaces, in order to utilise the lead as collector, and under the circumstances mentioned this is doubtless good practice.

At the *National* plant (S. Chicago) the refining skimmings still carry some antimony, and in order to recover this they are reduced together with the desilverised antimony skimmings in a reverberatory furnace with coke breeze, the product being sold as second-class hard lead.

The fact that the refining skimmings have been already freed from silver indicates that their return to the refinery blast furnace or to the softening furnace, with the consequent necessity for the subjection of the lead reduced from them again and again to the process of desilverisation, is illogical, but it is not easy to see a better way of handling this product.

**Refinery Blast Furnace.**—One blast furnace in a large plant is frequently set aside to receive in addition to the ordinary ore charge varying amounts of the following refinery bye-products—viz., copper dross from softening furnace, refining skimmings from refining furnace, litharge from cupel furnaces, slag from dross furnace (sweating cupels), old cupel tests and retorts, accretions from softening and refining furnaces. In addition to the ordinary ore-charge, from 12 to 25 per cent. of the above products can be added as an extra, this additional burden serving to increase rather than reduce the ore smelting capacity of the furnace. In addition to a heavy bullion-fall, a comparatively high-grade copper matte rich in silver is generally made in the refinery blast furnace, and ore-matte often forms part of the charge.

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## CHAPTER XV.

### THE PATTINSON PROCESS.

THIS process is based upon the discovery made by Pattinson in 1829 that the crystals which form during the slow cooling of molten lead are poorer in silver, while the still liquid remainder is richer in silver than the original lead. This circumstance is explained by the experiments of Reich,\* who found that small quantities of silver alloyed with lead lower its melting point until the silver contents rise to about 700 ozs. per ton ( $2\frac{1}{4}$  per cent.), above which critical point the higher melting point of silver exerts its influence, and subsequent additions of that metal raise the melting point of the alloy.

In the case of poor leads subjected to cooling from a molten condition, if two-thirds be removed as crystals these will be about one-half as rich as the original lead, or about one-quarter as rich in silver as the liquid remainder, which is ladled out into another pot and allowed to accumulate till there is enough for a repetition of the process, or lead of equal tenor from some other source is added. By a sufficient number of repetitions the liquid remainder from the last crystallisation is raised to about 500 to 600 ozs., when the process has to stop, the enriched lead being cupelled.

**Interference of Impurities.**—Most impurities of the work-lead interfere with the process, because they tend to hinder the formation of large crystals, besides producing a large quantity of dross.

*Arsenic*, *antimony*, and *tin* are especially deleterious, the former passing chiefly into the crystals, while the two latter are to some extent concentrated in the liquid lead. *Zinc*, also, if present (which, fortunately, is but rarely the case), is fatal, causing, even in small quantities, a reversal of the process, the silver going into the crystals and dross, as in the Parkes process. *Copper* is largely eliminated as pot dross, but to a slight extent becomes concentrated in the residual lead. *Bismuth* is very perfectly concentrated in the latter (as is also *nickel*); upon this fact is based the Freiberg process for its commercial utilisation. When the lead is practically free from As, Sb, and Cu, a simple “poling” in the pot in which it is melted down and removal of the dross is sufficient preliminary purification; but when these impurities are present it must be previously softened.

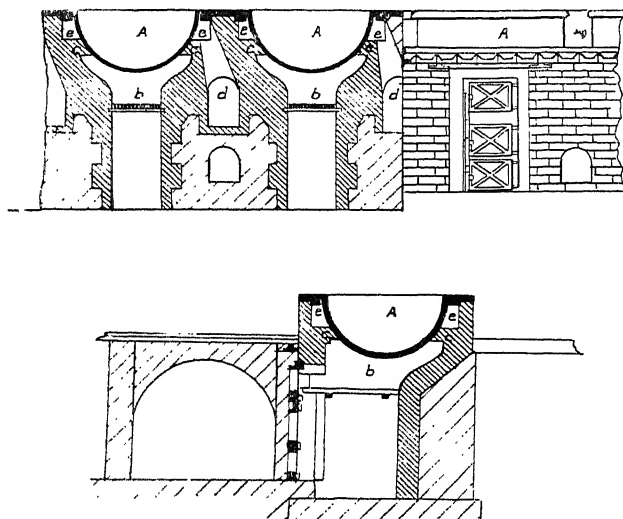
There are two main variations of the process—namely, *Hand and Steam Pattinsonisation*. In the former a series of eight to fifteen cast-iron pots is employed; into one of the central ones the lead is introduced and melted down; it is then cooled, with water and by constant stirring, until crystals form, which are ladled out with a perforated skimmer into the pot on the right, while the liquid lead is ladled into that on the left. In the latter

\* *Berg- u. Hüttenmannische Zeitung*, 1862, p. 251.

process, only two melting pots and a crystalliser are required, stirring is done with a jet of steam, and, instead of ladling out the crystals, the still liquid lead is drawn off into moulds from the bottom of the crystalliser. This process is an improvement on one now obsolete, in which mechanical stirrers were employed.

#### HAND PATTINSONISATION.

**Plant Employed.**—The number of the pots employed varies, according to the richness of the lead, from eight to fifteen, all of which are arranged in a row side by side to form a so-called “battery.” Part of such a “battery” of Pattinson pots is shown in Figs. 235 and 236, in which A A are the pots, *b b* the fireplaces, *c* a ring of brickwork which both helps to support the pot and separates the fireplace from the annular flue, *e*, round which the



Figs. 235 and 236.—Pattinson Pots.

flames circulate before passing by the downtakes into the flues, *d*. It will be seen that each pot is fired independently of its neighbours; all the pots are of the same size, except that on the extreme right, called the “market pot,” which, having only to receive the desilverised lead, is usually only two-thirds the size of the remainder. The crystals are removed from the pots, sometimes by perforated hand ladles, but more usually by larger ladles worked by small cranes.

**Mode of Working.**—Two methods are in use, called the method by thirds and that by eighths. In the former, two-thirds of the lead is allowed to crystallise and is removed to the right, and the remaining one-third of enriched lead, which is twice as rich as the original lead, or nearly four times as rich as the crystals, is passed to the left. In the method by eighths, seven-eighths of each pot is removed as crystals, and the remaining one-eighth of liquid alloy is three times as rich as the original lead. This system



requires only two to six pots, and is suited to the treatment of very low-grade leads, since the concentration in any given pot is more rapid; it was formerly in use near Stolberg (Rhen. Pruss.) for the treatment of lead carrying only 16 ozs. silver per ton, the resulting market lead containing 6 dwts. and the enriched lead 400 ozs. per ton. The method by eighths is, however, complicated and involves the accumulation of numerous heaps of crystals in various stages of concentration.

The method by thirds is the common mode and requires from ten to twelve, or even fifteen pots, according to the richness of the final lead produced. The bulk of the work-lead to be treated is melted down in the "charging-pot," but the position of this charging pot in the series varies with the proportion of silver in the lead. The pots are numbered in order from the market pot upwards, and the charging-pot may be, in a series of twelve pots, anywhere between No. 6 and No. 10. At each particular works the silver contents of each pot are kept as far as possible constant, and leads of very different grades may be melted down, each in that pot to which its silver contents most nearly approximate. The average silver contents of each pot at several different establishments is given in Table XLIV.

The pots are kept whitewashed, in order to prevent corrosion by the litharge which forms, as well as to prevent adhesion of the side crusts which collect on cooling. The lead being melted down in the charging-pot is "drossed" with a perforated skimmer, sawdust being sometimes added in order to facilitate the drying and collection of pasty skimmings, and, if no previous softening has taken place, a "poling" with green wood is appropriate at this stage. After skimming again, the fire is withdrawn, and transferred to some other pot which is in the melting-down stage, and water is sprayed on the bath from a hose. The crusts which form round the sides are broken off with a chisel-bar and pushed down again to be remelted, while the bath is continuously stirred in order that it may cool as uniformly as possible throughout. Crystals now form in the liquid, and are removed by the long-handled perforated ladle, which is lowered into the pot sideways, filled, and raised by the operator bearing down with his weight on the long handles, a greased roller fixed close to the pot on a swivel pivot serving as a fulcrum both for raising the ladle and for transferring its contents to the next pot. After allowing the ladleful of crystals to drain for a few minutes, an operation much facilitated by jerking two or three times on the handle, it is transferred to the next pot "down the house"—i.e., usually to the right—while after two-thirds of the charge has been thus removed, the remaining one-third of still liquid lead is ladled into the adjoining pot "up the house"—to the left. If there is lead on hand equal in silver contents with that in the pots to the right and left respectively, a similar operation can now be commenced with each of these simultaneously.

If only lead of one grade is being treated, two or more charges of original lead are carried through the series of pots "down the house," being crystallised in each pot successively before enough is accumulated for a crystallisation in the pot next above the charging pot. Generally speaking, at the commencement of any given campaign, all the pots on the right of the charging pot are about one-third full of lead, and those to the left are two-thirds full; the two-thirds of crystals which pass to the right, and the one-third

of liquid lead which passes to the left, then form charges for crystallisation both right and left all the way down and up the series. With 12-ton pots the additions of original lead will be sometimes 9 tons and sometimes only 36 cwt., according to the state of the charging-pot.

It is but rarely that lead of a single grade is treated in any given plant, and leads of higher or lower silver contents are added to that pot the richness of which most nearly corresponds to their own. Control of the working of the plant is kept by a daily or occasional assay of each pot; if it be found that the lead in the market-pot is getting somewhat too rich, the contents of each pot from the market- to the charging-pot are ladled back one step into the pot immediately above it.

The rich lead in the last pot on the left is often still further concentrated, after lading back the usual two-thirds as crystals into the pot below, by allowing it to cool still further, pressing the large perforated ladle into it so as to keep back the crystals, and then dipping out only the liquid portion which runs into the bowl. In this way the so-called "bottoms" may be as much as twice as rich as the crystals which are ladled back into the pot below.

In another modification of the process, five-sevenths are removed as crystals, and two-sevenths left as enriched lead. This is not very different to the ordinary method of thirds; but almost every works has some little peculiarity of manipulation in carrying out the process.

**Examples of Hand Pattinsonisation.**—The hand process has been, at several localities, replaced by the steam-stirring process, but it still survives in a few localities and notably at several English works, also at Linares, Freiberg, and one or two other places on the Continent of Europe.

At *Cordoba* (Spain)\* lead smelted from Linares ores and carrying 13 to 14 ozs. of silver is concentrated by thirds up to 56 ozs. in a battery of eleven pots, of which, however, only nine are generally in use. The pots are of 10 tons' capacity, and weigh 30 cwt. each; they are cast at a local foundry, and cost £20 apiece; their average life is from three to four years, except the charging-pot, which wears out much sooner owing to the harder usage. Counting from the left-hand end, Nos. 1 and 2 are usually idle with this poor lead, No. 3 is the market-pot, while No. 9 is the charging-pot, and No. 11 averages 56 ozs. of silver per ton. The lead melted down in the charging-pot averages only 14 ozs. of silver, but this is at once brought up by addition of the bottoms from the next pot on the left (No. 6, counting from the market-pot) to an average of 16 ozs., at which it is crystallised.

The battery is worked by two shifts of four men and one fireman, but the actual working hours, except for the fireman, do not average more than eight. Each shift melts down from 130 to 150 bars in the charging-pot, and brings out 75 to 80 bars of desilverised lead, besides casting a potful of silver-lead as it accumulates, about every other day. The coal consumption is about 46 per cent. by weight of the desilverised lead turned out. The total cost per ton of desilverised lead is about Ptas 5·00, or, say, 4s. per ton, which, however, does not include the cost of resmelting dross.

The pot dross made is large in amount, and it is liquated in two reverberatory furnaces, the lead produced being returned to the pots, and the liquated dross, which amounts to about 4·7 per cent. of the weight of the

\* *Private Notes*, 1908.

original lead and contains 60 to 65 per cent. lead, being added to the blast-furnace charge. Further particulars are given in Table XLIV.

At *Freiberg*,\* a special reason is found for retaining the Pattinson process in the bismuth contents of the work-lead, which averages 0·04 to 0·06 per cent. Bismuth, as already observed, becomes concentrated in the liquid lead, so that the original lead, containing from 133 to 257 ozs. of silver per ton, and 0·05 of bismuth, is concentrated in four crystallisations into a rich lead with 650 ozs. silver per ton and 0·17 per cent. bismuth. The impoverishment of the lead, however, is only carried to the point where it assays 33 ozs. silver, the further desilverisation of this poor lead, now containing less than 0·02 per cent. Bi, being carried out by means of the Parkes process. The recovery of the bismuth from the rich lead during the cupellation of the latter is referred to in Chapter xvii. It is found that by the combined process there is a saving of 18 to 20 per cent. on the total cost of desilverisation as formerly carried out by the Pattinson process alone, besides a better recovery of silver, the market lead now containing only  $4\frac{1}{2}$  dwts. as against a former minimum of 6 to 8 dwts.

**Cost of the Process.**—This varies at different localities from about 7s. up to 15s. per ton, according to the poverty or richness of the lead treated and to the cost of labour and materials; some detailed figures are given on Table XLIV. The amount of coal required averages 4 cwt. per ton of lead.

**Mechanical Pattinsonisation.**—This modification—using a mechanical stirrer in a deep cylindrical pot—was introduced in the sixties to do away with the very heavy labour involved in the hand process, which is only economical where the current rate of wages is very low. Its complexity seemed to more than offset the slight economy of labour effected; and though in successful use for some years at Rouen (France) and Stolberg (Germany), the process has been quite obsolete for the past twenty years. A good description is to be found in the work of Schnabel.†

#### STEAM PATTINSONISATION.

This improvement on the hand process, invented by Luce and Rozan at Marseilles, has largely displaced the older process, and is in use at Newcastle (Cookson's works), Marseilles, Przibram, Linares (La Tortilla), and several other places in Europe, as well as at Eureka (Nevada).

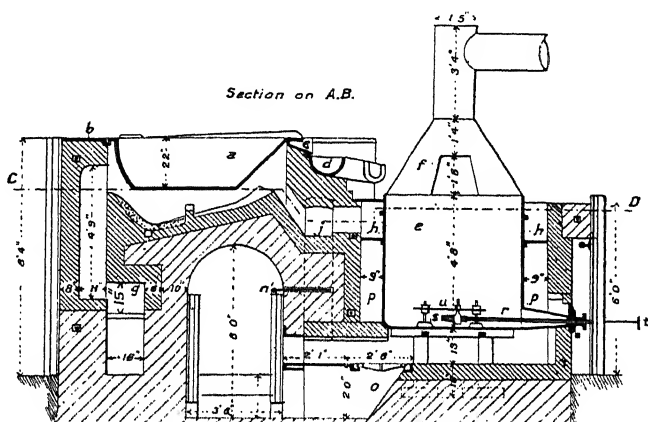
**Plant Employed.**—The apparatus used at all the above places consists of two melting pans, which are used alternately, and a crystalliser, besides large oval moulds for the liquid lead of each charge and a steam-crane to handle the ingots of lead. The plant at Przibram is shown at Figs. 237 and 238.‡ In these *a a* are the two cast-iron melting pans, each of which holding 7 tons of lead, can be raised by means of the crane, so as to pour its contents through the spout over the inclined plate, *c*, and the trough, *d*, into the cast-iron crystalliser, *e*, which holds 20 tons, and is covered with

\* *Oester. Zeits. f. B. u. H.-W.*, 1887, p. 421; *Berg. u. Hüttenm. Zeitung*, 1887, pp. 45, 192.

† *Op. cit.*, p. 570; see also Phillips, *Elements of Metallurgy*, 1891, p. 691, and Roswag, *Désargement des plombs argentifères*, 1884, p. 293.

‡ *Zdrahal, Oesterr. Jahrbuch*, vol. xxxviii., p. 1; also Hofman, *Metallurgy of Lead*, 1906, p. 418.

the wrought-iron hood and pipe, *f*, to carry off the escaping steam and fumes. Each melting pan is heated by a separate fireplace, *g*, the furnace gases passing either round the annular flue, *h*, which serves to heat the upper



to keep them hot, to the flue *l*. The furnace gases can also pass direct to the chimney through the flue *m*, their course being regulated by the dampers *n* and *n'*. The crystalliser itself is heated by a separate fireplace, *o*, the products of combustion circulating round the lower annular flue *p*, and thence passing to the flue *l*. The discharge spouts, *k*, have planed outside flanges, and are closed by means of slide valves working on these flanges

Fig. 239.—Plan of Bottom of Crystalliser and Steam Inlet.

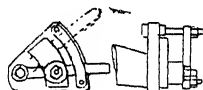
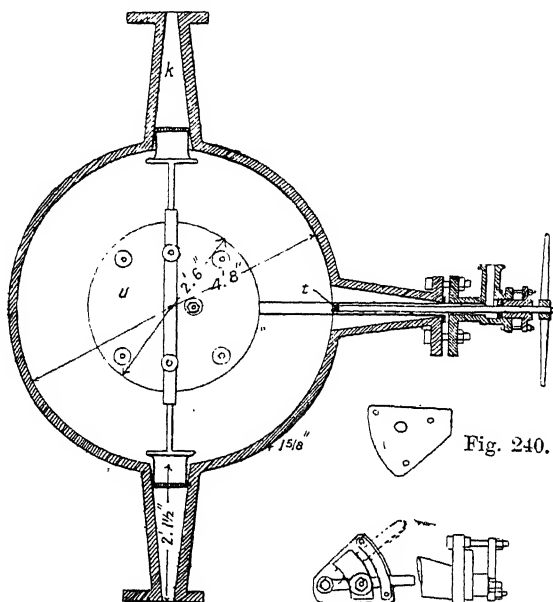


Fig. 241. Fig. 242.

Figs. 240 to 242.—Details of Slide Valve.

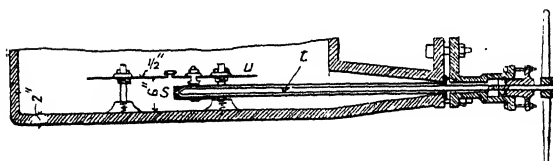


Fig. 243.—Section of Steam Inlet.

Figs. 239 to 243.—Details of Steam Pattinson Kettle.

and moved by hand levers, shown in Figs. 240 to 242; when opened, the molten lead runs through a movable trough into the moulds, *g*, which hold over 3 tons, and into each of which an iron hook is inserted before tapping out the lead to serve as a handle for the block.

The steam inlet consists of a steam pipe, *r*, screwed into a nozzle, *S*, provided with a plug valve, which is opened and closed by means of a rod,

*t*, running through the pipe, and screwed in or out of the nozzle. The steam issuing from the nozzle, *S*, is distributed through the lead by means of the baffle plate, *u*. Figs. 239 and 243, which show details of the steam inlet and bottom of the crystalliser, are self-explanatory.

**Mode of Working.**—The lead is melted down in one of the melting pans, which takes six hours, drossed, and run into the previously heated crystallising pot by raising the back end of the pan with the crane. Once in the crystalliser, the fire underneath is withdrawn, and the discharge spouts warmed up; steam is turned on for two minutes, and the lead drossed through the doors in the hood. The bath is stirred by means of a jet of dry steam at 45 lbs. pressure, and cooled by spraying a little water from jets near the top of the hood, taking care to inject only a little at a time to avoid explosions. After fifteen or twenty minutes the steam is shut off, while the crusts which have formed round the sides are barred off through the doors in the hood, and the splashes of lead on the hood are scraped down. This shutting off of steam is repeated about twice, and after about three-quarters of an hour two-thirds of the lead has crystallised. The steam valves are so arranged that this point is indicated simply by the resistance to the passage of the steam being so great as to prevent it from passing up through the lead, so that the cessation of boiling in the lead-bath indicates the completion of the operation.

The steam is then shut off, the slide valves opened, and the still liquid lead run into the moulds with their iron eyes. The fire under the crystalliser is then replaced, and the remaining two-thirds of crystals (say,  $13\frac{1}{2}$  tons) is remelted, which takes about three hours; in the meantime  $6\frac{1}{2}$  tons more lead, with the same silver contents as that in the crystalliser, have been melted down in one of the melting pans, so that as soon as the crystals are remelted it can be run in and a new operation begun. The other melting pan contains lead equal in value to the crystals which it is expected to produce from the next operation, which is being melted down in readiness for the crystallisation next following. In this way each complete operation on 20 tons of lead takes four hours, all the work being done by two men per shift. The consumption of fuel for the whole operation is at Przibram 35 per cent. by weight of the lead charged.

The products from 100 tons of previously liquated lead are (at Przibram) as follows:—

TABLE XLIII.

Products.	Weight from 100 lbs. Work-lead.	Percentage of Total Lead Present.	Percentage of Total Silver Present.
Rich lead, . . . . .	42·99	42·58	97·36
Desilverised lead, . . . . .	44·76	45·01	0·17
Scrap lead, . . . . .	3·02	3·03	0·61
Dross, . . . . .	9·94	8·04	1·45
Flue-dust, . . . . .	0·46	0·35	0·07
Loss, . . . . .	..	0·99	0·34
Total, . . . . .	101·17	100·00	100·00

TABLE XLIV.—EXAMPLES OF THE PATINSON PROCESS.

Reference,	1	2	3	4	Newcastle-on-Tyne.		7	8	Eureka.
	Hand.	Hand.	Hand.	Hand.	Hand.	Steam.	Steam.	Steam.	
Modification of process employed,	21 ozs.	96½ ozs.	35 ozs.	14 ozs.	30 ozs.	40 ozs.	160 ozs.	246 ozs.	
Silver contents of bulk of lead treated,	..	10	15	10	..	13-16	20	50	
Capacity of pots, tons,	..	9½ dwts.	6½ dwts.	18 dwts.	..	9 dwts.	9½ dwts.	1 oz. 84 dwts.	
Assay of market pot No. 1,	18 dwts.	19 dwts.	..	2 ozs.	..	..	19 dwts.	2¼ ozs.	
Assay of contents pot No. 2,	2 ozs.	2 ozs.	..	3¼ ozs.	..	..	2 ozs.	5½ ozs.	
" " No. 3,	4½ ozs.	4 ozs.	..	5 ozs.	..	..	3½ ozs.	10 ozs.	
" " No. 4,	11¼ ozs.	7¼ ozs.	..	8 ozs.	..	..	7 ozs.	20 ozs.	
" " No. 5,	*21½ ozs.	12¾ ozs.	..	*16 ozs.	..	..	11½ ozs.	33½ ozs.	
" " No. 6,	41 ozs.	22½ ozs.	..	33 ozs.	..	..	21 ozs.	55 ozs.	
" " No. 7,	79½ ozs.	37 ozs.	..	56 ozs.	..	..	45¾ ozs.	105 ozs.	
" " No. 8,	129 ozs.	59½ ozs.	*35 ozs.	..	..	..	32¼ ozs.	84 ozs.	
" " No. 9,	220 ozs.	*96½ ozs.	..	..	..	..	62 ozs.	157½ ozs.	
" " No. 10,	..	167 ozs.	..	..	..	..	103¼ ozs.	*246 ozs.	
" " No. 11,	..	273 ozs.	..	..	..	..	*100 ozs.	..	
" " No. 12,	..	..	288 ozs.	..	..	523 ozs.	274 ozs.	515 ozs.	
" " No. 14,	358 ozs.	514 ozs.	..	..	..	..	..	..	
" liquid " bottoms " of last pot,	..	..	12-62 t.	..	..	..	42-99 t.	..	
Weight of rich lead from 100 tons work-lead,	..	..	84-21 t.	..	..	45-0	44-76 t.	..	
" poor "	35	45	45	46	..	12	35	..	
Consumption of coal per cent.,	..	..	3-2	1½	..	2-1	1-0	..	
Loss in lead per cent.,	..	..	+ 0-02	..	..	— 1-5	— 0-34	..	
Gain or loss of silver per cent.,	..	..	6-0	1: 6½	9: 5	10: 11½	..	..	
Cost per ton s.d., labour,	..	..	2-6½	4: 2¾	0: 9½	4: 9½	..	..	
" " fuel,	..	..	2-9	0: 10¾	0: 2½	4: 11	..	..	
" " castings, tools, repairs,	..	..	..	..	..	..	..	..	
Total,	..	..	11: 3¾	6: 7¾	10: 5	20: 8	..	..	

Notes.—The charging-pot is marked by an asterisk; all silver contents are given per ton of 2,240 lbs.

References.—1 and 2. Phillips, *Elements of Metallurgy*, 1891, p. 690. 3. Teichmann, *Preussische Zeitung*, vol. xv., p. 44. 4. *Priv. Notes*, 1908. 5. Roswag, *Deargentation des Plombs Argentifères*, p. 286. 6. *Ibid.*, p. 300. 7. Schenabel, *Handbuch der Metallhüttenkunde*, vol. i., p. 527. 8. Zdrhal, *Oesterreichische Jahrbuch*, vol. xxxiv., p. 1. 9. Curtis, *Mon. U.S. Geol. Survey*, vol. vii., 1884, p. 163.

At *Eureka* (Nev.) the crystalliser holds 50 tons of lead, and the moulds, into which the liquid lead is poured by a movable spout run on wheels, like that shown in Fig. 233, though enormously larger, holding in fact 3 tons each. At this works the lead is only enriched to about 550 ozs., as the subsequent enrichment is so slow and imperfect that it is not considered worth while bringing it any higher.

At *Przibram*\* the plant consists of three kettles, and the desilverised lead with only traces of silver still carries traces of Sb and Sn, for which reason it is refined in two 26-ton reverberatories, the operation lasting twenty-four hours, after which the lead is tapped and cast into moulds. It is again remelted in 4.4-ton kettles, and cast into pigs for market.

At *La Tortilla* (Linares)† there are two 10-ton crystallisers, each with its pair of melting pans, and the lead treated is of much the same grade as that treated by the hand process at Cordoba—viz., 14 to 16 ozs. of silver. The desilverised lead at these works is mostly worked up into sheet pipe and shot, for all of which there is a ready market.

In Table XLIV. the value of each separate crystallisation at *Przibram* and *Eureka* is given, corresponding with the separate pot-assays of the hand process.

Compared with the hand process, the Luce-Rozan modification has only the disadvantages of a greater capital outlay, and greater cost of repairs and renewals. These are more than offset by the advantages, viz.:—Saving of two-thirds of the labour, three-fifths of the fuel, and two-thirds of the dross produced in the old process. Another great advantage for dealing with comparatively impure lead is that the preliminary softening, so necessary for the ordinary hand-worked process, can be dispensed with, for the oxides produced by the steam from the traces of antimony, copper, &c., present are largely volatilised and carried away by the current of steam to be caught in suitable condensers, while so long as these impurities are present the lead itself is to a great extent protected from oxidation. Inasmuch, moreover, as steam-refining purifies lead more perfectly than air-softening, the resulting product is softer and sells somewhat better.

*Tredinnick's*‡ improvement on the Luce-Rozan process consists in the use of a series of 50-ton crystallising kettles (twelve in number for 300 ozs. bullion), each of which is supported on a hydraulic ram, so that it can be raised at will, thereby enabling the liquid lead after crystallising to be run into the adjacent kettle. Heating is carried out by means of a brick-lined jacket round the kettle, in which burning natural or producer gas is circulated. Each crystalliser has two outlets, that on the left for running out the residual liquid lead is double, and has a strainer to prevent escape of the crystals, while that on the right, for running out the crystals after remelting, has a free run without a strainer. The average assay values at which it is estimated to be possible to maintain the series of pots is as follows:— $\frac{1}{2}$  oz., 1 oz., 2 ozs., 4 ozs., 7 ozs., 12 ozs., 20 ozs., 35 ozs., 60 ozs., 100 ozs., 180 ozs., 300 ozs. The cost of treatment is estimated to be per single operation:—

\* *Mineral Industry*, vol. xv., p. 534.

† *Private Notes*, 1908.

‡ *Austin, Min. and Scientific Press*, Jan. 19, 1907, p. 89.



	\$	s. d.
Fuel gas (natural gas), 10,000 cubic feet at 5 cents per 1,000,	0.50	2 1
Labour for one operation, $\frac{1}{2}$ of 3 men at \$3 each, . . . . .	0.75	3 1 $\frac{1}{2}$
Superintendence (\$6 per day for 36 operations), . . . . .	0.17	0 8 $\frac{1}{2}$
Steam for crystallising, . . . . .	0.05	0 2 $\frac{1}{2}$
Tools and repairs, . . . . .	0.10	0 5
General expenses, . . . . .	0.20	0 10
Total per operation, . . . . .	<u>1.77</u>	<u>7 4<math>\frac{1}{2}</math></u>

With the original hand Pattinson process using 10 to 15-ton pots only three to four operations are usually performed in twelve hours; with the Luce-Rozan or steam-Pattinson process using 50 ton kettles five operations may be performed per kettle in twelve hours; whereas with the Tredinnick process using 50-ton kettles it is expected that about twelve operations can be performed in an 8-hour shift at, say, 40 minutes per operation. Even should each operation last on an average an hour all told there would still be a large saving.

A plant on this principle is now being introduced at the *Omaha* works of the Am. Sm. and Ref. Co.

**Bye-products.**—The bye-products of the Pattinson process, besides scrap lead which is as far as possible returned to that part of the process to which it belongs, are only drosses of various degrees of richness, which are frequently collected by means of sawdust sprinkled on the surface of the pots. They are treated by running down in a reverberatory lead furnace with the addition of fine coal; the dross lead thus produced contains most of the impurities of the original lead, and has to be softened before submitting it to a repetition of the process. The average amount of pot-dross and skimmings produced during the Pattinsonisation of poor lead with only 8 ozs. of silver per ton, may be estimated at 20 per cent. by weight; with richer lead the proportion may reach 25 to 30 per cent. The actual *loss* of lead, however, is always small, varying from, perhaps,  $\frac{1}{2}$  per cent. up to 2, or, perhaps, 3 per cent., according to the degree of purity. A curious bye-product, sometimes met with in the steam Pattinson process, though in small quantity, is an incrustation of minium, which forms in the crystallisers and is evidently produced directly by the action of the steam upon the lead at a temperature just below redness.

## CHAPTER XVI.

## THE PARKES PROCESS.

**Influence of Other Metals.**—This process, foreshadowed by Karsten, but only made practicable in 1852 by Parkes, is based upon two facts—viz., (a) the greater affinity of silver for zinc than for lead; and (b) the insolubility of zinc-silver alloys in lead which is already saturated with zinc. The process of desilverisation with zinc consists, in brief, of stirring 1 to 2 per cent. of that metal into a bath of molten work-lead, heated to above the melting point of zinc ( $415^{\circ}\text{C.}$ ), and allowing it to cool, when a crust or scum rises to the surface, containing nearly all the silver. A repetition of the process with a smaller quantity of zinc produces a lead which is practically free from silver, containing usually not more than about 4 to 6 dwts. per ton, as against 9 to 15 dwts. by the Pattinson process.

Besides *silver*, zinc has an affinity for various other metals, and in particular for *gold* and *copper*, with both of which it combines before taking up silver. Gold is perfectly extracted by the first addition of zinc, and copper enters the zinc crust almost as readily, though not quite so completely. It is noticeable that a gold-copper crust, containing all but the minutest trace of the gold present, is formed long before the lead begins to take up zinc to any large extent, the gold-zinc and copper-zinc alloys being apparently almost insoluble in zinc-free lead; whereas it is impossible to produce a silver crust without first saturating the lead at the given temperature. Seeing that *copper* alloys so readily with zinc, it is important to purify the bullion as completely as possible by softening and repeated drossing before commencing the desilverisation; since, otherwise, a lot of poor copper crust is formed before any silver at all can be extracted, thus consuming zinc and wasting time.\*

*Bismuth* does not at all interfere with the desilverisation, but *antimony*, in the proportion of 0.1 per cent., and *arsenic* in even smaller proportion, not only retard the rising of the crusts, but actually prevent a clean separation from the underlying lead; so that it becomes impossible to turn out, with only three zincings, market lead of only 6 dwts. silver from work-lead containing upwards of 0.1 per cent. antimony, or 0.05 per cent. arsenic. Fortunately, arsenic is much more readily oxidised than antimony, so that it is completely removed after only eight or ten hours' softening, and the question practically resolves itself into the removal of antimony. *Nickel* and *cobalt* (if present) would, like copper, enter the crust, but, fortunately, both metals are easily removed by drossing, besides which they are of rare occurrence.

\* v. Experiments by Kirchoff in *Metallurgical Review*, vol. i., p. 224; which are also quoted by Egleston, *Metallurgy of Silver, Gold, and Mercury in the United States*, vol. i., p. 94.

Impurities in the zinc itself, such as iron, arsenic, copper, sulphur, tin, &c., have the same unfavourable effect as if they were present in the lead; therefore, only the best procurable brands of commercial spelter should be used. This fact was first made clear by the experiments of Kirchhoff already referred to.\*

**Plant Employed.**—Lead bullion is always desilverised in cast-iron "pots," "kettles," or "pans," as they are variously called, which are arranged beneath the level of the softening furnaces, and above that of the refining furnaces, or kettles, so as to enable the lead to be run from one part of the process to another without undue handling. In some of the older works on the Continent of Europe, Pattinson plants have been made to serve for the Parkes process, with as little alteration as possible, and in these the lead frequently has to be pumped from one part of the plant to another, but in most English and American works the plant is arranged in terrace form, as shown in Fig. 244,† which is a section of the refinery at *Monteponi* (Sardinia), and may be regarded as typical of a plant using furnaces for refining. Fig. 245 is a sketch-plan and section of the larger refinery of the

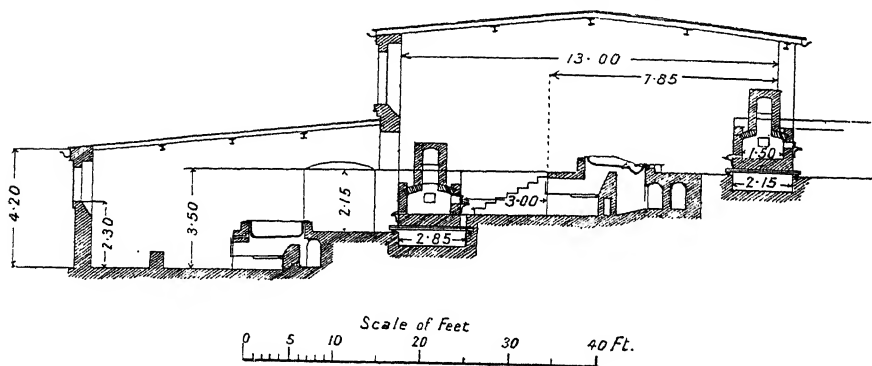


Fig. 244.—Section through Lead Refinery (Monteponi).

Broken Hill Proprietary Co. at *Port Pirie* as it appeared some years ago, in which kettles are employed for refining and furnaces for liquation, while softening is performed in two stages. In this figure *a* represents the receiving or unloading platform, *b* is the so-called "copper," or drossing furnace, *c c* are "antimony" or softening furnaces, *d d* the desilverisation pots, and *e* the liquation furnace for zinc crusts; *f f* are refining kettles, *g* is the casting pot, *h* the dross liquating furnace, and *i* the refinery blast furnace for drosses, skimmings, litharges, &c.; *j* and *k* are sidings for unloading and loading respectively. The retorting and cupellation rooms are situated beyond the crust liquating furnace, in what is practically a separate building.

The pots must be perfectly smooth inside, and are always of cast iron unless they have to serve for subsequent refining of the lead, when cast steel is sometimes employed, as already mentioned.‡ Pots which have to serve for the whole process from drossing to refining, as at *Lautenthal* (Harz) and

\* v. Also Föhr, *Berg- u. H. Zeitung*, 1888, p. 28; also Egleston, *op. cit.*, p. 93.

† Ferraris, *E. and M. J.*, Oct. 25, 1905, p. 783.

‡ v. the last Chapter.

*Tarnowitz* (Silesia), are made small ( $12\frac{1}{2}$  to 15 tons); those for desilverisation pure and simple are made to hold from 30 to 50 tons, and may last for 150 to

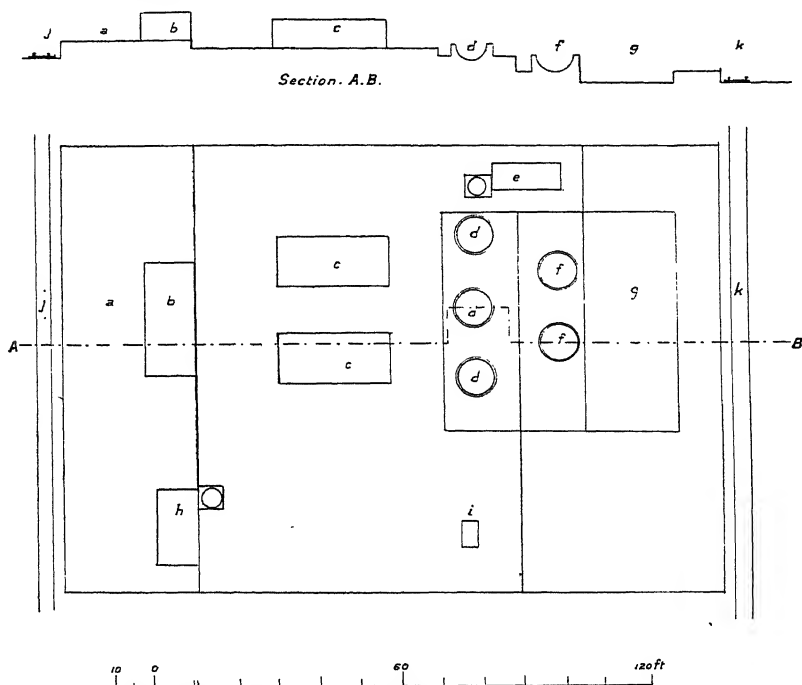


Fig. 245.—Sketch-plan and Section of Port Pirie Refinery.

300 charges. The form is usually a segment of a sphere, and the pot is supported either by a rib cast round it at half its depth, or simply by its flange, or by a brick ledge, as in Fig. 246, which shows a common way of supporting a desilverisation pot above the fireplace, by means of which the temperature of the lead is raised prior to stirring in the zinc in order to melt the latter and incorporate it thoroughly.

A 30-ton pot is frequently made 8 feet 10 inches to 9 feet wide by 3 feet 4 inches to 3 feet 6 inches deep (the radius being 4 feet 7 inches), and has a flange from 6 to 9 inches wide. The 35-ton pots at *Port Pirie* are 9 feet diameter by 3 feet 9 inches deep with a 6-inch flange, and when full would hold 45 tons, although

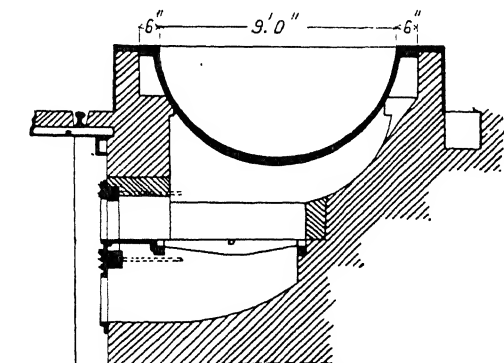


Fig. 246.—Desilverisation Kettle.

the charge actually worked in them is only about 33 tons. Larger pots are sometimes made elliptical, like those of the Omaha and Grant refining works at *Omaha* (Neb.), which are about 12 feet long by 7 feet wide and 3 feet 3 inches deep, holding 48 tons; but the 50-ton pots of the *Balbach* (N.J.) and *Pittsburg* works are round, like the 40- to 45-ton pots of the *Howdon* (Newcastle), *Pertusola* (Italy), *Peñarroya* (Spain), and other works.

**Amount of Zinc Required.**—The quantity of zinc required to extract the silver varies according to the richness of the bullion treated, though, proportionately, poor lead takes more than rich. Every refinery has its own table of quantities, and Roswag\* has attempted to deduce general formulæ adapted to bullion of any given tenor, which do not, however, seem to have commended themselves to practical refiners. At Freiberg, Plattner† found that the following quantities are required:—

Quantity of Silver in Base Bullion.		Percentage of Zinc Required.
Per cent.	Ozs. per Ton of 2,240 Lbs.	
0.0963	31.46	1.34
0.3825	124.92	1.84
0.5080	165.94	1.96
0.8400	256.79	2.45

But much less is found sufficient in most modern refineries by using the crust over again, so as more completely to saturate the zinc contained in it. Hofman‡ gives a table by Eurich, which is the basis of most tables used in modern works, but in practice a fourth zining is dispensed with save in cases where, for some reason, the lead retains accidentally too high a tenor of silver after the third zining.

Reference has been made to the fact that the gold-copper crust is formed without saturating the lead-bath with zinc. A very small proportion of zinc is, therefore, sufficient for the first or "gold-zinc" addition, as little as 0.10 to 0.12 per cent. by weight being found sufficient for properly softened bullion containing not over 8 dwts. gold per ton,§ and subsequent additions are directed solely to the removal of silver. When the bullion contains a large quantity of gold, and is practically free from copper, no separate gold crust is made, but, in that case, all the silver produced from the silver crust has to be parted for gold, and it is far preferable to extract the gold first in as concentrated a form as possible, provided the gold contents of the lead are sufficiently low. Under conditions prevailing in Australian and at a good many North American works, however, such a large amount of gold is found in the work-lead that it is impossible to remove the whole of it with one zining, and in such cases it is often considered preferable to make only one class of crust and part all the silver.

When a gold crust is first taken off, the additional zinc, for bullion of 300 ozs. and upwards per ton, is occasionally added in three portions (viz.,

\* *La désargentation des plombs argentifères*, Paris, 1884, p. 251.

† *Berg- u. Hüttenm. Zeitung*, 1889, p. 117.

‡ *Op. cit.*, 1906, p. 449.

§ *Private Notes*, Port Pirie, 1896.

two-thirds, one-fourth, and one-twelfth respectively of the total); but with ordinary bullion of 150 to 300 ozs. per ton, the work is almost always done with two silver-zincings of about equal amount, say 0·8 to 1·0 per cent. each of the weight of lead under treatment, of which only the second is fresh zinc. Low-grade bullion of less than 125 ozs., if sufficiently softened, can be perfectly desilverised with only two zincings, yielding one gold and one silver crust. The first silver crust from rich bullion should contain from 2,000 to 4,000 ozs. per ton at least, and leave the lead with 30 to 90 ozs. per ton; the second crust will run from 200 to 700 ozs., and leave a very poor lead; while, if a third addition is required at all, the resulting crust will contain only an ounce or two of silver.

In any case only the first crust is liquated; those obtained from subsequent zincings contain much free zinc and are returned to the pot as the first zincing of the next following charge.

Separation of the crust is much facilitated by stirring into the bath 1 lb. of sal ammoniac to each 10 tons of work-lead. This addition, long since tried by Rössler and Edelmann, has been patented in the U.S.A. by Raht; \* its functions would seem to be the absorption of dross, cleansing the metallic surfaces, and prevention of oxidation.

Process of "Zincing."—The pot, having been whitewashed and heated up to the temperature at which a splinter of wood will ignite, is filled with lead by tapping from the softening furnace direct, or by syphoning from a "collecting pot" in which it has been drossed, if such is employed. When it is tapped direct from the furnace a gutter of angle-iron is employed, which empties into an iron pipe reaching to the bottom of the pot, so as to produce as little oxidised dross as possible. In American works the lead is usually drossed at once after addition of any liquated lead from gold crusts of previous charges. Unless, however, a collecting pot has been employed, it is much better to allow the lead to cool in the pot, so as to skim off as much copper dross as possible before commencing desilverisation, as this plan permits the complete extraction of gold in an abnormally small weight of gold-crust. A common practice, which secures this end, is that known as "ringing"; the lead tapped from the softening furnace into the desilverising pot is allowed to cool there quietly till a "ring" of solid metal from 4 to 6 inches wide has solidified all round the edge. This ring is chipped off and returned to the softening furnace, the lead is stirred and again skimmed, and then and not before is the firing started strongly in order to get the lead hot enough for the first zincing, which requires a temperature of not less than 450° C. The zinc is usually employed in slabs which are often simply melted down on top of the lead and stirred in by hand. At some works, however, it is put in a perforated iron box, which is held at the bottom of the pot by an iron rod until the zinc has melted out of it in little streams which can be easily mixed with the lead; at others, again, it is melted separately and poured into the lead, the object in both cases being to avoid oxidation.

However it is melted down, the zinc must be very thoroughly stirred in, which takes thirty or forty minutes; this may be done in three ways:—

**Hand Stirring** is done by means of a circular perforated sheet-iron disc ("paddle") attached to a 6-foot handle with a crosspiece. Two men on

\* U.S. patent, 826,114, July 17, 1906.

opposite sides of the pot lower their paddles down the sides and raise them in the middle, so producing an upward current in the centre of the lead-bath; the operation is most fatiguing, and many attempts have been made to replace it by steam stirring, which, however, has the grave disadvantage of producing a more oxidised and poorer crust; on this account a majority of the best and most modern works still adhere to hand stirring.

**Steam Stirring** is done by means of a steam pipe reaching down vertically some 2 feet into the centre of the pot. The steam must be perfectly dry in order to avoid explosions, and, therefore, steam traps for condensed water are provided; and the pipe must be thoroughly warmed up before lowering it into the pot. At, or a little above, the melting point of zinc ( $415^{\circ}\text{C.}$ ) there is comparatively little oxidation of that metal, though at temperatures much above  $450^{\circ}$  (at a cherry red for instance) oxidation is rapid. As soon as the steam is turned on it produces a downward current in the centre of the bath, but the bubbles drive away the molten zinc together with the incipient crusts to the sides, whence they are continually pushed toward the centre by a wooden hoe.

**Mechanical Stirring.**—The *Howard stirrer*, invented and first tried at the *Pueblo S. and R. Co.'s* works in Pueblo, is now largely used all over the world; its construction is shown in Fig. 217. It comprises a stout cover resting on the top of the pot so as to exclude the air, a horizontal screw propeller attached to a vertical shaft passing through the centre of the cover and a stout frame resting upon it supporting bevel gearing which drives the propeller at 110 revolutions per minute, the gearing itself being driven either by a rope-drive, as shown in Fig. 217, or by a small steam or compressed-air engine or electric motor, the power required being about 5 H.P. The whole apparatus is hung by means of blocks from a traveller, which runs upon the same rails used for the Howard Press to be described later.

The more thorough stirring possible with this appliance, and the avoidance of oxidation by means of the cover, together facilitate the separation of a richer crust, which is treated to greater advantage in the refert, yielding a larger proportion of clean bullion and zinc and less bye products.

When the "stirring in" is completed, or somewhat before, the fire is damped down with wet slack and the doors opened so as to cool the pot, and, if steam has been used for stirring, it is blown through the lead a little longer with the same object. After about two or two and a half hours the surface crust formed begins to adhere to the sides of the pot, and is then removed by means of a perforated skimmer with a long gaspipe handle worked upon the edges of the pot as a fulcrum, or suspended by a hook which serves the same purpose. At the *Pueblo Smelting and Refining* works (Pueblo, Colo.) the Howard skimmer\* is used, which consists of two semi-circular perforated plates hinged in the centre, like a "butterfly valve," and folding downwards. In this condition they are lowered into the lead; they are then unfolded just under the crust, and raised with it above the level of the bath to drain. The crust must be thoroughly drained and shaken free from lead before turning into the liquating kettle or into a cast-iron mould. That part which adheres to the sides of the pot is then barred off with a long steel chisel, and the operation of skimming repeated till no more

\* Hofman, *Mineral Industry*, vol. II, 1893, p. 40, and *Metallurgy of Lead*, 1906, p. 473.

crusts form. About eight samples of one A.T. are then taken and cupelled; the silver beads are added together and parted to see if all the gold has been removed.

In the meantime the pot is heated up, liquated lead from the first silver crust of the preceding charge is added, and, as soon as hot enough, the first silver-zinc is stirred in. This consists of the second (and third, if any) silver crust from a previous operation which contains much free zinc, and a little fresh zinc, if necessary, to make up the amount required. The operation of collecting and removing the silver crust is exactly similar to that described for the gold crust; the crust is generally poured and pressed into moulds, so as to be conveniently handled and charged into the liquation furnace.

The lead, now containing from 30 up to 90 ozs. silver, is again sampled and treated with zinc for a second silver crust, which is also generally moulded,

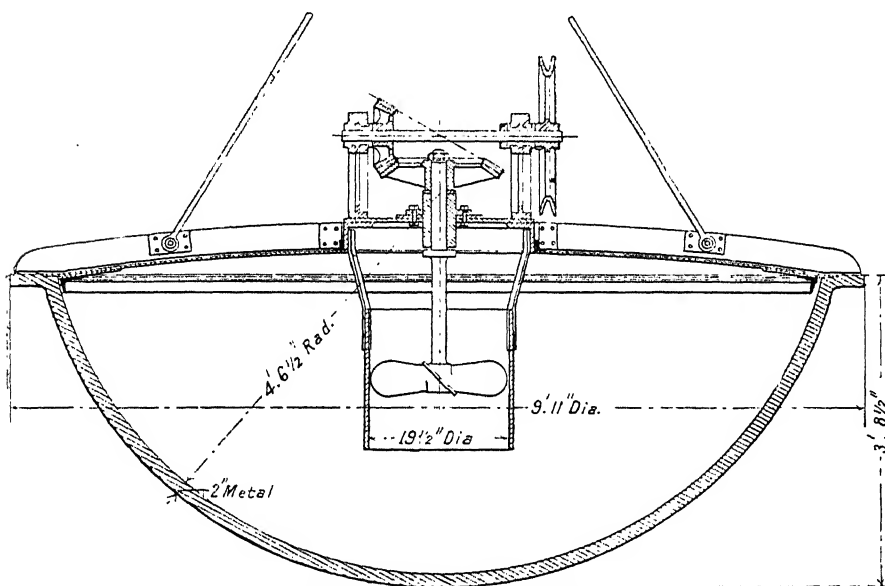


Fig. 247.—The Howard Stirrer.

or simply deposited on an iron plate close to the pot to wait for the next charge. If, after removing this second silver crust, the lead still contains  $\frac{1}{2}$  oz. or so of silver per ton, the necessity for a third zincing can be avoided by blowing steam through the pot for half an hour or more, which causes a small additional formation of zinc crust carrying silver, and reduces the silver contents of the lead to 3 to 4 dwts.

**Time Required and Fuel Consumed.**—The time required for each melting and stirring averages half an hour to an hour, cooling two to three hours, removal of crusts half an hour to an hour, reheating one and a-half hours to two hours, so that five or six hours may be reckoned as the average time for each complete zincing. With three zincings, giving one gold and two silver crusts, ordinary bullion can be desilverised in about fifteen hours,



including running the lead into the pot and syphoning it off afterwards. The *Port Pirie* practice, subsequently described, of doing the work very slowly has, however, many advantages. Bullion which is comparatively poor in silver can be desilverised somewhat more quickly, as the quantity of crust to be removed is less.

The consumption of coal for desilverising and liquating the crusts varies from  $2\frac{1}{2}$  to 3 per cent. of the weight of the original base-bullion softened.\* At some of the refining works in Chicago, *crude petroleum oil*, sprayed by means of compressed air, is used as fuel, and is said to be more economical than coal, as well as affording the convenience of being extinguished and relighted in a moment, which accelerates both the cooling and the reheating. Natural gas has also been used at Pittsburg with the same objects and similar results.

**Examples of Desilverising Practice.**—At *Freiberg*, lead which has been already desilverised by the Pattinson process and still contains 0.1 per cent. Ag ( $32\frac{1}{2}$  ozs. per ton), is desilverised in 20-ton kettles with three additions of zinc, together amounting to 1.485 per cent. of the weight of lead. The first addition carries off the gold and brings down the silver contents to 8 ozs., while the second reduces it to below 13 dwts., and sometimes to as little as 6 dwts., in which case the third addition is dispensed with.

At one of the refineries in Mexico † where a separate gold zincing is always given, an example of the composition of the lead after each zincing is as follows:—

Original assay of work-lead, . . . . .	170.15 ozs.
Assay of pot after gold zincing, . . . . .	133.9 "
"    "    1st silver zincing, . . . . .	40.4 "
"    "    2nd    "    "    . . . . .	6.7 "
"    "    3rd    "    "    . . . . .	0.26 "

At *Peñarroya* (Spain) ‡ the refinery is on the, now general, terrace plan, and is worked in 12-hour cycles, 45-ton softening furnaces and pots being employed. The work-lead containing only 42 to 45 ozs. of silver with no gold, two zincings are found sufficient. The first consists of the second crust from the preceding operation together with 230 kilos. of fresh zinc. The second consists of 270 kilos. of fresh zinc, and the crust obtained is all saved to be returned to the next following charge. The first crust, which is liquated in kettles, contains 8 to 9 per cent. silver (2,600 to 2,900 ozs. per ton), and, in spite of softening at a low temperature, no less than 5 to 6 per cent. of copper, which interferes greatly with the subsequent distillation and other processes. Analysis of the softened lead, moreover, shows 0.006 to 0.008 per cent. copper, which is too high for clean work in desilverising such low-grade bullion, and it is probable that by very carefully liquating down the whole of the lead at a low temperature in a furnace, instead of in a drossing kettle, it could be better freed from copper, and consequently a smaller proportion of this obnoxious element would make its way into the silver crust.

\* According to Hofman the average is about 54 lbs. per short ton = 2.7 per cent., by weight.

† *Private Notes*, 1902.

‡ *Private Notes*, 1908.

At the *Port Pirie* refinery,\* where 500 tons of lead per week are desilverised, the plant consists of four desilverising pots of 30 tons' capacity, and the bullion treated averages 300 to 330 ozs. of silver, with about 5 dwts. of gold per ton. Only three zincings are given, and the last of these leaves the lead with only about 6 dwts. of silver per ton. The lead is tapped direct from the softening furnace into the zincing pot, where it is allowed to cool for three hours in order to admit of a further drossing, which much facilitates the clean separation of the crusts, especially of the gold crust, and also facilitates its subsequent retorting. The amount of "pot dross" removed averages about 200 lbs., containing 210 ozs. Ag, 96 per cent. Pb, and 0·2 per cent. Cu, which is treated together with that produced in the first drossing furnace.

The temperature is then raised, and the *gold-zinc* stirred in. In the first instance, on starting a new campaign a single addition of 75 lbs. of zinc is used, and is found to extract all the gold. The crust obtained from this very thoroughly softened bullion is found to be still capable of withdrawing most of the gold from a second charge of lead, although the total amount of zinc added is not more than one-tenth the amount required to saturate the lead. It is, therefore, returned to the next potful of lead ready for treatment, from which nearly the whole of the gold is extracted, and the enriched (because twice used) gold crust is set aside until sufficient accumulates for a charge in the liquation furnace. This second pot, however, still retains a trace of gold, so that a second addition of 50 lbs. of fresh zinc has to be stirred in to clean it. The resulting skimmings, which contain most of the added zinc still free, serves as first addition to the next following charge, so that each charge is treated twice for gold—viz., first with once used zinc skimmings, and then with 50 lbs. of fresh zinc. In this way only 50 lbs. of fresh zinc are required for each charge of lead, and a rich gold crust is obtained from poor lead at the expense of only a slight delay.

The perfection of the extraction of gold with this exceedingly small quantity of zinc (only 0·07 per cent.) is remarkable, the lead retaining only about 4 grains per ton, or, say, 1 oz. per ton of silver contained in it, and the *Port Pirie* practice would seem to be a great improvement on the American practice described by Hofman,† in which 250 to 300 lbs. of zinc are required to obtain a gold crust from bullion of the same richness in gold. It depends entirely, however, upon a much more thorough drossing and softening than is usual in America, and the only possible objection, the greater time it requires, is obviated by the simple expedient of distributing an amount of work usually performed in two desilverising pots over a plant of three, or the work of three over four pots, without any increase in the labour, or in wear and tear of pots, and with practically no increase in the consumption of fuel.

After removal of the second or intermediate gold crust, the lead is again heated up, and the first silver-zinc stirred in; this consists of some 3 or 4 tons of lead-zinc-silver alloy with about 10 per cent. Zn, and about 700 ozs. Ag per ton, obtained as the second silver crust from the previous charge cast into moulds, and stacked by the side of the pots. This is stirred in very thoroughly by hand, and the pot is allowed to cool, the crusts which then rise being skimmed off by hand, drained over the pot, and cast in moulds. The whole process of heating, stirring in, cooling, and removing the principal

\* *Private Notes*, 1896.

† *Op. cit.*, 1906, p. 448.

silver crust takes about eight hours, the full plant of four pots being worked in a leisurely manner, cooling very thoroughly, by a gang of five or six men; this plan has been found to give the best results without consuming any more fuel.

When no more crust rises, the lead (now carrying 90 ozs. per ton) is reheated, and the second silver zinc, consisting of 825 lbs. of fresh zinc, stirred in. After cooling and skimming the second crust, which is moulded and stacked for the next charge, the lead rarely contains above 6 dwts. Ag per ton. It is at once syphoned off by the ordinary Steitz syphon into the refining pots below. Occasionally, when the lead has been less thoroughly softened than usual, it still retains, after the second zincing, from  $\frac{1}{2}$  oz. to 1 oz. of silver, and in such cases a third zincing is given, using a small quantity only of fresh zinc, which is stirred in and removed at the lowest possible temperature so as to clean the lead as thoroughly as may be.

The crusts average, after liquation, gold crust 750 ozs. Ag, 13 ozs. Au, 4 per cent. Zn; silver crust 3,000 ozs. Ag, 20 per cent. Zn. The pots used at the works (and cast locally) last, on an average, during the treatment of about 150 charges. The consumption of fuel is about 27 cwt. of coal per charge treated, or, say, about 3.4 per cent. of the weight of the original base bullion. The total amount of zinc used is 1.34 per cent.

The practice above described in detail is that in vogue at Port Pirie a few years ago. According to Delprat,\* however, in more recent practice one single charge of 200 lbs. of zinc is used three times over—that is, on three successive charges of lead—before being withdrawn for treatment as gold skimmings, after which it contains 6 ozs. of gold, 200 ozs. of silver, 10 per cent. of zinc, and only 80 per cent. of lead. The Howard press† is in use for squeezing these and all other skimmings.

The latest practice of gold zincing at Port Pirie is described in detail by Bayly‡ and being unique is worthy of repetition here. Each potful of lead requires for gold extraction from 80 to 100 lbs. of fresh zinc, which is passed through either two or three further charges (making in all three or four passages through fresh quantities of lead) in accordance with the scheme shown in Table XI.A. The pots are worked in pairs A and B, and the table shows the gold zincing of fourteen successive charges alternately in either pot in successive cycles of five charges each. Each charge of lead is zinced twice for gold, the zincings being numbered 1 and 2, the second being always fresh zinc (generally 100 lbs.), while the first zincing is a variable amount of a partially saturated crust which has been through the pot a varying number of times. Zinc which has been once through the pot is called "2nd first," the crust which has passed through a second charge after the first (making two in all) is called "1st time through," and that which has passed through two charges besides the first (making three in all) is called "2nd time through," whilst after the next passage it is called "gold press." With this explanation, the way in which the various additions of zinc pass through successive charges of lead will be readily understood from the table. Each five charges, with a total addition of 400 to 500 lbs. of fresh zinc turn out a gold crust, and this quantity of zinc has been through either three or four successive potful of lead as shown—extracting the very

\* *Trans. Aust. I.M.E.*, vol. xv., 1907, pp. 1-29.

† Described in this Chapter, p. 416.

‡ *Trans. Aust. I.M.E.*, vol. xv., 1907, pp. 80, 81, 82.

small quantity of gold present with great perfection, as indicated by the assays of the contents of the pots after various numbers of zincings.

TABLE XLV.—GOLD-ZINCING AT PORT PIRIE.

POT	NUMBER OF		ADDITION CONSISTS OF				GOLD PRESS
	CHARGE	ADDITION	FRESH ZINC	SECOND FIRSTS	1st. TIME THROUGH	2nd. TIME THROUGH	
A	1	1 <sup>ST</sup> 2 <sup>ND</sup>	2			1	→
B	2	1 <sup>ST</sup> 2 <sup>ND</sup>	2	1			
A	3	1 <sup>ST</sup> 2 <sup>ND</sup>	2	1			
B	4	1 <sup>ST</sup> 2 <sup>ND</sup>	2		1		
A	5	1 <sup>ST</sup> 2 <sup>ND</sup>	2			1	→
B	6	1 <sup>ST</sup> 2 <sup>ND</sup>	2	1			
A	7	1 <sup>ST</sup> 2 <sup>ND</sup>	2	1			
B	8	1 <sup>ST</sup> 2 <sup>ND</sup>	2	1			
A	9	1 <sup>ST</sup> 2 <sup>ND</sup>	2		1		
B	10	1 <sup>ST</sup> 2 <sup>ND</sup>	2			1	→
A	11	1 <sup>ST</sup> 2 <sup>ND</sup>	2	1			
B	12	1 <sup>ST</sup> 2 <sup>ND</sup>	2				
A	13	1 <sup>ST</sup> 2 <sup>ND</sup>	2	1			
B	14	1 <sup>ST</sup> 2 <sup>ND</sup>	2		1		
A	15	1 <sup>ST</sup> 2 <sup>ND</sup>	2			1	→

The two gold-zincings for each pot-charge take about seven hours each—viz., heating up  $1\frac{1}{2}$  hours, stirring-in  $\frac{1}{2}$  hour, cooling  $4\frac{1}{2}$  hours, skimming  $\frac{1}{2}$  hour—which leaves from 1 to  $1\frac{1}{2}$  hours out of the 8-hour shift for sampling. The two silver-zincings which follow take the full 8 hours each, requiring more time for heating up in order to dissolve the larger quantity of zinc and consequently for cooling also.

At first, on starting a new cycle, the 1st silver zincing consists of 500 lbs. of fresh zinc, which makes the "2nd press." To clean the lead an addition is then made of 734 lbs. of fresh zinc, which when skimmed off is called "3rd skim," and is set aside to be passed through the other pot of the pair, after which it becomes "2nd skim" or "2nd press." The scheme of working is shown in the following Table XLVI., from which it will be seen that each pot-charge is zincd twice and produces one silver skim for the press. The lead in the pot is now assayed, and if it contain more than 6 or 7 dwts. per ton another small zincing is given and skimmed at as low a temperature

as possible. The total weight of skimming removed from each pot is about 3 tons.

TABLE XLVI.—SILVER-ZINCING AT PORT PIRIE.

Pot.	Number of		Addition Consists of		Second Skim —i.e., Silver-Press.
	Charge.	Addition.	Fresh Zinc.	Third Skim.	
A	1	{ 1st	2	1	→
		2nd			
B	2	{ 1st	2	1	→
		2nd			
A	3	{ 1st	2	1	→
		2nd			
B	4	{ 1st	2	1	→
		2nd			

The average composition of the contents of the pots at Port Pirie after each zincing is as follows :—

	Original Softened Lead.	Gold Zincing.		Silver Zincing.	
		After 1st Zinc.	After 2nd Firsts.	After 2nd Zinc.	After 3rd Zinc.
Au, per ton, .	0·3 oz.	19 grs.	tr. to $\frac{1}{2}$ gr.	none	none
Ag, ozs. per ton,	70-80	70-80	68-75	20-30	0·3 to 0·4
Zn, per cent., .	..	..	..	..	0·56
Cu, „ .	0·12	..	..	..	·00026
Sb, „ .	0·089	..	..	..	·00325

The coal consumption is about 3·2 per cent. by weight during the whole process of desilverisation, which lasts 32 hours. Various details of the practice are given in Table XLVII. side by side with similar data from other plants.

At the works of the *St. Louis Smelting and Refining Co.*,\* Collinsville, Ills., soft lead from Missouri with only 2 to 3 ozs. per ton is now being desilverised down to  $2\frac{1}{2}$  dwts. per ton with one zincing, the silver recovered just paying the cost of the operation, while the profit lies in the higher quality of the lead produced, which, being freed from the traces of Cu, Sb, and Ni that accompanied it, now fetches a higher price.

With regard to the extraction of the gold contents of work-lead in a separate gold-crust, it should be noted that this practice is only possible when the amount of gold present is small. With bullion rich in gold, say 5 ozs. per ton or over, such as is produced regularly at some of the Australian and occasionally at some American works, it is not possible with one zincing to remove the gold contents without removing at the same time a considerable proportion of the silver, and without leaving behind at the same time

\* *E. and M. J.*, March 16, 1907, p. 518.

so much gold as to make it worth while to part the silver resulting from the first silver-crust—the limit for which, under Australian conditions, is, according to Blakemore,\* about 15 dwts. Au in every 1,000 ozs. of Ag. In such a case it is simpler to make only doré bullion—that is, to treat all the crusts together and to part the whole of the silver produced. Since the great increase in the amount of gold contained in their furnace bullion, dating from about fifteen years ago, American refiners have also to a great extent discontinued making a separate gold-crust. At the *Germania* works,† when a separate gold-crust was made gold losses began to appear in the refining plant, which disappeared again as soon as the separate gold-crust was abandoned and all the silver bullion was parted for gold. It seems probable that successful extraction of gold in a separate gold-crust is only possible:—

(1) When the lead bullion contains not over 1 oz. of gold for each 250 ozs. of silver contained in it.

(2) When the dressing has been so thoroughly done that a gold-crust can be obtained with only a small amount of zinc; if copper be present in the work-lead to any considerable extent, proper separation of a small gold-crust is impossible.

Some further data of desilverising practice at various localities are given in the following table:—

TABLE XLVII.—COMPARATIVE DATA FROM DESILVERISING PLANTS.

Reference, . . . .	Lautenthal, Upper Harz.	Tarnowitz, Upper Silesia.	Freiberg, Saxony.	Port Pirie, S. Australia.		Dapto, N.S.W.
	1	2	3	1896	1906	6
Charge in desilverising pots, . . . tons,	12½	13½	20	30	33	11
Silver contents of lead, ozs. per long ton,	49	16	33	330	70-80	131
Weight of first zincing, . . . lbs.,	49	..	220	75	80-100	200
“ second “ . . . .	110	..	165	825	750	220
“ third “ . . . .	180	..	88			
Total weight of zinc employed, per cent.,	1·21	1·00	1·00	1·34	1·15	1·59
Value of gold crust, Au, . . ozs. per ton,	..	..	5	13	6	200-300
“ “ Ag, . . . .	..	..	1486	1000	200	1200-2000
Value of first silver crust, Ag, . . . .	..	..	..	3750	3000	1000-2000
“ second “ . . . .	..	..	..	700	150-200	..
Silver contents, lead after first zincing, ozs.,	48	..	8	305	70-80	5-20
“ “ second “ . . . .	6	..	13 dwts	90	20-30	0·50-2·00
“ “ third “ dwts.,	3	2	4	6	6·8	..
Total time in desilverisation pans, hours,	15½	22	20	22-24	32	..
Coal consumption, . . . per cent.,	9·76	7·4	..	3	3·2	3
Gold contents, original lead, . . dwts.,	..	..	2½	6	6	280
Percentage of gold crust produced, . . . .	..	..	..	0·91	..	10%
“ silver crust produced, . . . .	6	..	2·25	..	2·3 %	9%
Life of pot, charges, . . . .	..	..	..	150	..	60

- References.—1 and 2. Schnabel, *Handbuch der Metallurgie*, vol. i., pp. 540 and 541.  
 3. Plattner, *Oesterr. Zeitung f. Berg- und Hüttenwesen*, 1887, p. 421. 4. *Private notes*, 1896.  
 5. Delprat, *Trans. Aust. I.M.E.*, vol. xii., 1907, pp. 1-29; and Bayly, *ibid.*, pp. 80, *et seq.*  
 6. Blakemore, *Trans. Aust. I.M.E.*, 1898, vol. v., pp. 221, *et seq.*

\* *Trans. Aust. Inst. M.E.*, 1898, vol. v., p. 221.

† *Min. Ind.*, vol. xvi., 1907, p. 680.

**Composition of Zinc Crusts.**—Before liquation, it is difficult to determine the average composition of these, owing to the lead which they retain mechanically; even after liquation some free lead still remains, being prevented from running together by films of zinc, lead, and other oxides between the globules. Besides gold, silver, copper, iron, &c., *tellurium*, if present, becomes concentrated in the silver crust,\* whence part of it becomes volatilised during the operation of retorting as telluride of zinc, the remainder finding its way into the retort bullion, and, finally, into the last litharges, where no doubt it exists as a tellurite of lead.

The following table shows some analyses of zinc crusts.

**Use of Aluminium-Zinc.**† —Not only is the lead contained in the zinc crust prevented from liquating out perfectly by the films of zinc and lead oxides, but the perfect collection of the zinc-silver alloy is also impeded by these same films, and this is the principal reason why it is not possible to desilverise with a single addition of zinc. At temperatures up to 500° the lead will not dissolve sufficient zinc to carry up all the silver present, and if it be attempted to raise the temperature to 700° with the idea of dissolving the whole quantity of zinc at one operation, so much oxide, both of lead and zinc, is formed that liquation is impossible, and a great deal of the silver remains in solution instead of going into the crust. Any addition, therefore, which tends to prevent oxidation of the lead-bath during the progress of desilverisation will facilitate the operation and yield a richer crust. Such a substance was found by Rössler and Edelmann‡ in aluminium,  $\frac{1}{2}$  per cent. of which completely prevents oxidation of zinc at the ordinary desilverising temperature, while even the amount of 0·007 per cent. Al by weight in the lead-bath which is being desilverised—if free from copper and arsenic, and very low in antimony—practically protects it from oxidation, the surface remaining bright and metallic.

*Antimony*, in the proportion of 0·1 per cent. or less, does not interfere; 1 per cent., however, interferes very much with the separation, besides causing a high zinc consumption. In the presence of so little as 0·1 per cent. of *copper*, however, or 0·05 per cent. of *arsenic*, the protective influence of aluminium disappears, and oxidation takes place, with consequent incomplete separation of the crust from the lead. All lead bullion, therefore, which is not thoroughly softened requires to be further purified by means of a copper-zincing before adding the aluminium-zinc alloy.

In practice, the Al-Zn alloy, with  $\frac{1}{2}$  per cent. Al, is produced by melting the zinc and stirring in the aluminium; this molten alloy, in quantity sufficient to extract all the silver present, is poured into the lead-bath, previously freed from copper and arsenic and thoroughly stirred at a temperature somewhat over 500° C. As the bath cools, two successive crusts are removed, the first (called the “cream crust”) containing most of the oxidised material, and resembling ordinary crusts in appearance; while the second, though richer in metallic lead, is almost free from oxides, and carries most of the silver.

At *Hoboken*, near Antwerp,§ where this process is carried out on base

\* Heberlein, *Berg- u. H. Zeitung*, 1895, p. 41.

† *E. and M. J.*, April 4, May 16, 1891, also Sept. 2 and Dec. 12, 1893.

‡ *Ibid.*

§ Schnabel, *Handbuch der Metallhüttenkunde*, vol. i., p. 576; also *E. and M. J.*, Sept. 2 and Dec. 12, 1893.

TABLE XLVIII.—COMPOSITION OF LIQUATED ZINC CRUSTS.

	Reference.	Silver Crusts.								Gold Crusts.	
		Altenau, 1880.	Lautenthal, 1880.	Mechernich, 1884.	Freiberg.	Globe, Denver, 1896.	Bleyberg, Belgium.	Hoboken, Antwerp.	Port Pirie.	Hoboken, Antwerp.	Port Pirie, S.A.
		1	2	3	4	5	6	7	8	9	10
Pb,	.	75.675	77.820	45.48	53.20	67.0	86.0	65.75	68	75.80	85.0
Zn,	.	11.78	12.11	35.45	39.70	13.2	10.5	15.20	17	10.15	4.0
Cu,	.	1.12	0.82	2.3	2.68	4.4	0.607	.	.	.	5.5
Ag,	.	1.855	2.42	15.17	4.55	9.96	17.25	8.10	13	2.4	.
As,	.	tr.	tr.	.	.	.	0.045	.	.	.	.
Sb,	.	.	tr.	0.36	.	.	0.40	.	.	.	.
Cd,	.	tr.	tr.	.	.	.	tr.	.	.	.	.
Ni,	.	tr.	tr.	.	.	.	tr.	.	.	.	.
Fe,	.	.	.	1.28	.	.	tr.	.	.	.	.
PbO,	.	4.75	4.00	.	.	.	.	.	.	.	.
ZnO,	.	0.60	0.44	.	.	.	.	.	.	.	.
Bi <sub>2</sub> O <sub>3</sub> ,	.	0.72	0.37	.	.	.	.	.	.	.	.
Sb <sub>2</sub> O <sub>3</sub> ,	.	0.63	0.98	.	.	.	.	.	.	.	.
Fe <sub>2</sub> O <sub>3</sub> ,	.	1.87	1.04	.	.	.	.	.	.	.	.
Ag, oza. per ton,	.	696	778	482.547	.	3215	.	.	4300	.	200
Au,	"	.	.	.	4.92	3.31	.	.	.	6½.8	6

References.—1 and 2. *Zeitsch. f. d. B. H. u. S. W. i. P.*, vol. xxxviii., p. 262. 3 and 4. *Ibid.*, vol. xxxiv., p. 92. 5. *Private Notes*, 1896. 6. Firket, *Ann. des Mines de Belgique*, 1901, pp. 237, et seq.; *Min. Ind.*, 1903, p. 427. 7 and 9. *Ibid.* 8 and 10. *Private Notes*, 1897.



bullion varying in richness between 500 and 3000 ozs., the "cream" or first portion of the crust is from 1 to 5 per cent. of the weight of lead treated, the second portion being from 5 to 9 per cent. by weight, according to the silver contents of the bullion. Both crusts are liquated separately, and, after giving up about 76 to 80 per cent. of their weight as liquated lead, the residues are melted down together, yielding a rich *zinc-silver alloy* and an *oxidised crust*. The former only is perfectly melted; it has a silver-grey colour, a fine-grained fracture, and is free from the iridescent oxidation colours seen in ordinary zinc crusts. It contains 20 to 35 per cent. Ag,  $3\frac{1}{2}$  to 5 per cent. Pb,  $1\frac{1}{2}$  to  $4\frac{1}{2}$  per cent. Cu, and 60 to 70 per cent. Zn, besides traces of Al, Sb, As, and Fe; the oxidised crust contains 2 to 6 per cent. Ag only. The higher percentages of silver above given are only found when treating rich bullion.

When treating low grade *Mazarron* bullion, averaging 49 ozs. of silver per ton, which is first softened by steam in the same pot in which desilverisation takes place, the total consumption of zinc is 4.42 per cent. Of this amount, 20 per cent. (0.29 per cent. on the lead) is added at first, and a gold and copper crust removed; then the remainder (4.13 per cent. on the lead), previously alloyed by melting with 0.5 per cent. Al, is poured in after heating the bath. On this poor lead the "cream" crust amounts to  $1\frac{1}{2}$  per cent. only by weight, and yields on liquation 62 per cent. of lead, 4 per cent. of rich alloy, and 34 per cent. of oxidised crust; the second, or "rich," crust yields 87.7 per cent. of lead, 9.8 per cent. of rich alloy, and only 2.5 per cent. of oxidised crust.

The weight of each kind of product obtained from 1000 parts of unsoftened or original base bullion, and the percentage proportions of silver and lead contained in each are given in the following table:—

TABLE XLIX.—PRODUCTS OF DESILVERISATION WITH ALUMINIUM ZINC.

Products.	Weight of zinc consumed.	Proportion of silver.	
		Lead.	Silver.
Refined lead, . . . . .	300.7	300.7	Trace
Softening dross, . . . . .	1.4	1.08	1.3
Skimmings and pot dross, . . . . .	4.6	4.08	Trace
Copper crust, . . . . .	2.9	2.18	9.9
Rich silver-zinc alloy, . . . . .	0.6	0.02	78.9
Oxidised crust, . . . . .	0.7	0.001	9.9
Loss, . . . . .		1.688	None
	1000.0	1000.001	100.0

According to Roswag, the average loss in the ordinary desilverisation process without aluminium is 1.60 per cent. of the lead, and 0.3 per cent. of the *true* silver contents of the lead; most schemeries, of course, show

\* Adapted from Schnabel, *op. cit.*, p. 377.

† Approximations only.

an overplus of silver caused by loss in the ordinary commercial assay of the base bullion.

Aluminium-zinc is also used in the refineries at Bleyberg, and at other places in Belgium and Germany.

#### TREATMENT OF ZINC CRUSTS AND ALLOYS.

As withdrawn from the pot zinc "crusts" or "skimmings" contain not only a large quantity of lead alloyed with the zinc, but also an excess of lead mechanically entangled in the spongy mass, and prior to the further treatment of the crust this entrained lead must be got rid of, which is done either by liquation in kettles or in furnaces, or by squeezing in the Howard press.

**Liquation in Kettles or Pans.**—In this system each desilverising pot has by its side two smaller and shallower pots.\* The American liquation kettle is always made with a convex bottom, as shown in Fig. 248, but on the Continent the form shown in Fig. 248a, is often used.† The liquation pot, of whatever form, must be heated very gradually, otherwise some of the zinc crust will be re-dissolved in the liquated lead. When no more lead exudes, that which is liquated is tapped into the liquated lead pot set below, and the crust is withdrawn and broken up, while still hot and brittle, into small pieces for subsequent treatment. This is done by bruising and cutting it with a shovel, or by forcing it in a pasty condition through an iron grating.

The liquated lead amounts to from 40 to 60 per cent. by weight of the crust charged; that from the gold crust assays 100 to 250 ozs. of silver per ton, and is added to the desilverising pot before the gold-zinc of the next charge; that from the silver crust assays 30 to 40 ozs., and is added to the next charge before the first silver-zinc. At *Freiberg* the liquated lead from a crust containing 1,200 ozs. of silver, which leaves the pot lead with 8 ozs. of Ag, runs itself only 11 ozs. It may be stated, as a general rule, that when liquation is performed carefully, whether in pots or in furnaces, the liquated lead will run from 30 to 40 per cent. higher in silver than that left in the pot from which the crust was removed.

Seeing, moreover, that in kettle liquation the crust has to float on a bath of lead, it is obvious that with the slightest want of care, or overheating, some of the precious metal contents of the crust will be redissolved by the underlying lead. In order to overcome this objection at *Monteponi* a shallow liquating pan has been introduced, the bottom of which has a gradual slope towards a smaller kettle in which the liquated lead collects, the connection between the two being always open. The steady drawing away of the lead as soon as melted, and before it can redissolve the crust, answers so well that from a dry liquated crust running from 1,600 to 2,600 ozs. of silver the liquated lead will usually contain less than 15 ozs.‡

\* Hofman, *op. cit.*, 1906, p. 453.

† Schnabel, *op. cit.*, vol. i, p. 544.

‡ Ferraris, *E. and M. J.*, Oct. 25, 1905, p. 733.

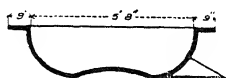


Fig. 248.

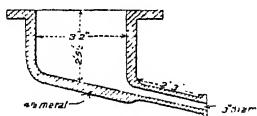
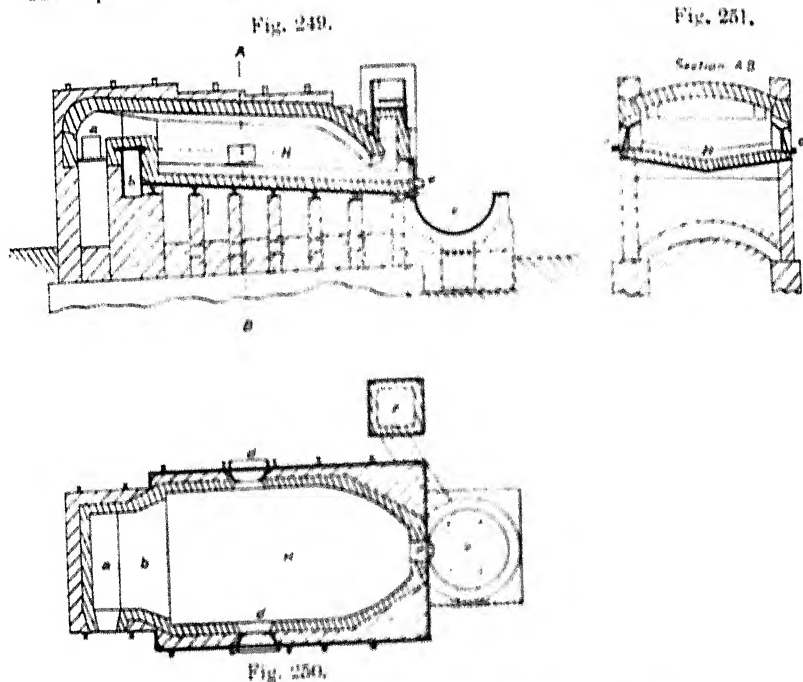


Fig. 248a.

Figs. 248 and 248a.—Liquation Kettles.

**Liquation in Furnaces.**—Although convenient for small works, at large establishments where the weight of crusts to be handled is great, liquation kettles may be advantageously replaced by furnaces, which treat all the crusts from several desilverisation pots. This simplifies the work at the latter, besides giving more uniform results. The reverberatory has the further advantage that a reducing flame can be employed, so avoiding the partial oxidation which takes place in the pots; and, when not over-driven, it extracts from the crust a larger proportion of poor lead, so yielding a richer liquated crust. The hearth of a liquating furnace may be simply a V or channel-shaped cast-iron plate, but more commonly a half-brick lining is built upon the plate to protect it.

The liquating furnace formerly in use at *Port Pirie* (S.A.) is shown in

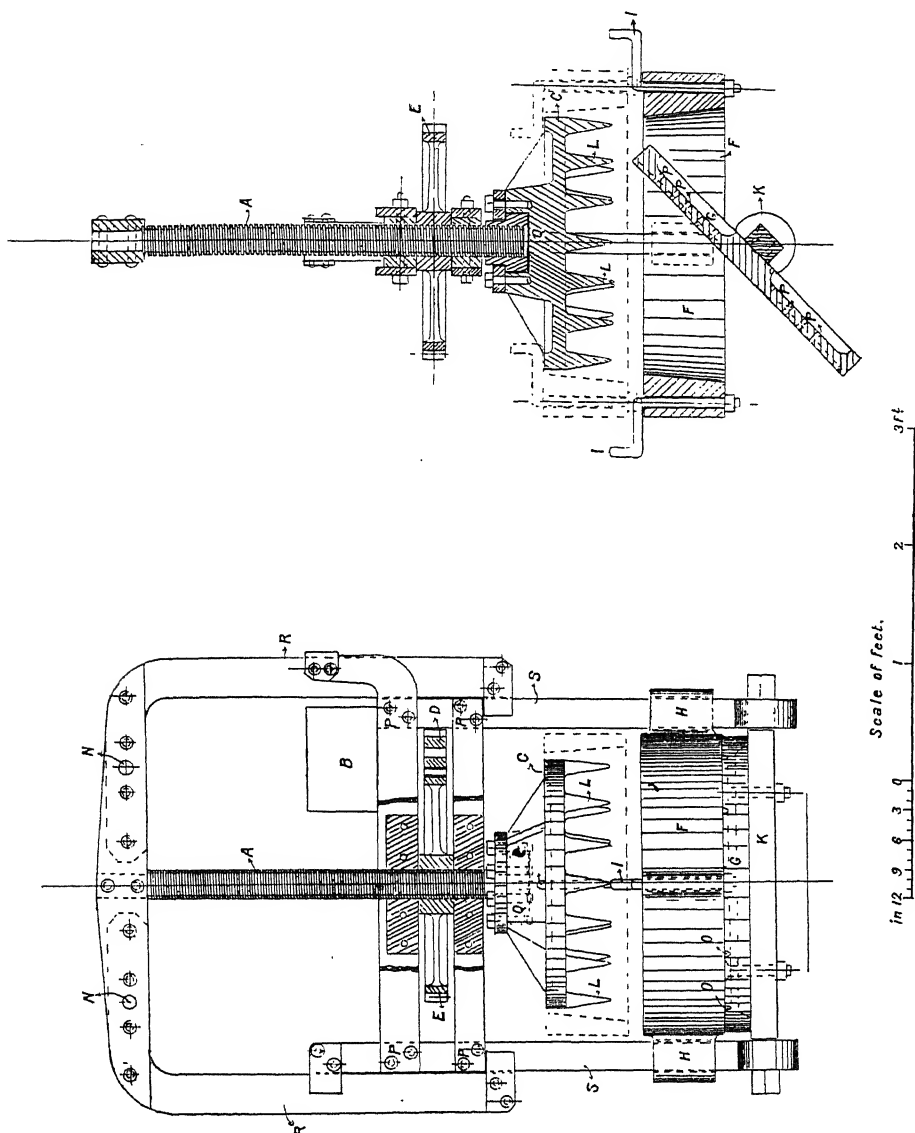


Figs. 249 to 251.—Crust Liquation Furnace (*Port Pirie*).

Figs. 249 to 251, in which *a* is the fireplace, *b* the bridge, with an air-cooled casting in the centre, and *H* the trough-shaped hearth, composed of a single course of firebrick inclined towards the centre and lengthwise towards the lead kettle *c*. The whole hearth is built in a wrought-iron pan, which rests upon a course of old rails supported by brick pillars, so as to be air cooled, and there are two working doors, *d*, besides the taphole opening, *e*. The same down-cast flue, *f*, serves for both the furnace itself and for the auxiliary fire beneath the kettle, in which the liquated lead is kept but so as to be ladled when convenient.

The charge of this furnace is about  $1\frac{1}{2}$  tons of zinc crust, which, with the

help of constant rabbling and a smoky flame of uniform temperature, is liquated in four hours, the lead accumulating in the lower part of the hearth, from which it is tapped out about twice a shift into the kettle. At first



Figs. 252 and 253.—Howard's Improved Alloy Press.

the charge is deposited chiefly at the lower or cooler end of the hearth, and is only drawn upwards nearer the fire as it loses lead and becomes more infusible. The daily capacity of the furnace is about 9 tons of crust, the

output of liquated lead is about one-third of the weight of crust charged, and its silver contents average, as already stated, about 40 per cent. more than that of the lead left in the pot from which the crust was removed; thus, at *Port Pirie*, the liquated lead from the silver crust averages about 120 to 130 ozs., while that from the gold crust averages only about 210 ozs. per ton. Gold crusts generally lose less weight in liquation than silver crusts, being already drier to start with. Great care must be taken not to overheat the crust, since overheating drives gold and silver back into the lead. The coal consumption in liquation is not generally accounted for separately; it averages less than  $\frac{1}{2}$  per cent. on the weight of bullion treated.

**The Howard Press.\***—This ingenious appliance, which has revolutionised the handling of zinc crusts, is shown in Figs. 252 and 253. It consists of a cylinder, F, tapered upwards, a perforated bottom, G, pivoted on the horizontal shaft, K, the ends of which rest in sockets in the uprights, SS, allowing the bottom to turn around the axis of the shaft. By means of the lugs, HH, the cylinder is movable along the uprights SS. At B is a motor, air, steam or electric, driving a pinion, D, which engages with the spur wheel, E, into which is threaded the screw, A. The whole frame, R R, which, with its contained mechanism, is slung from NN hangs vertically from a traveller, and when the motor is started the alloy cylinder and uprights, SS, with the motor and gears all move upwards vertically against the screw, A, with its toothed plunger, C, and along the guides, P.

When the cylinder is full of skimming and by the starting of the motor is brought up against the toothed plunger, the teeth sink into the skimming, and so press out the liquid lead and facilitate its escape through the perforated bottom. The motor is then reversed and the cylinder is run down, the toothed plunger is given part of a turn, so as to put the teeth in a different position, and the motor is again started upwards, making a series of holes in the mass of crust. By repeating the process several times most of the free lead in the crust is expelled and the residue becomes quite dry. When the pressing is over the cylinder can be hooked on to the plunger by the hooks, I, and on reversing the motor the compressed cake follows the bottom, which can then be tilted so as to dump it where desired.

The repeated puncturing of the mass of crust by the teeth of the plunger in a variety of positions serves not only to facilitate the escape of the lead, and so produce a very dry crust, but also to facilitate the breaking of it up into small pieces, which is necessary before it can be retorted.

#### Treatment of Liquated Zinc Crusts.

The original method of Parkes (1852-59) was to distil off the zinc in an ordinary Belgian furnace, but the retorts would not stand, and the process had to be abandoned for a time. Of late years, however, chiefly owing to the introduction of plumbago retorts and of improved furnaces, this process has again come into vogue and has, to a great extent, displaced other methods, except in a few localities. Three principal processes may be distinguished, the object in each being to separate the zinc and leave the silver in a rich lead suitable for cupellation.

\* *Private Notes*, 1902.

1. **Flach's** process of smelting the liquated crusts with ferruginous slags in a cupola and driving off the zinc into the slag and flue-dust.
2. **Corduries'** process of oxidising the zinc, together with part of the lead, by means of steam at a red heat. Zinc may be recovered as basic carbonate by means of Schnabel's process.
3. **Parkes'** process of distilling off the crusts and recovering most of the zinc as metal.

Besides the above processes for the treatment of ordinary zinc crusts with from 4 to 13 per cent. of silver, two special processes, applicable only to the treatment of the rich zinc-silver alloy of the Rössler aluminium-zinc process, must also be briefly referred to, viz. :—

4. The **sulphuric acid** process ; and
5. The **electrolytic** process, in which the zinc is recovered as very pure metal.

1. **Flach's Process.**—This process was in use many years ago at several works in England and on the Continent, including the *Mechnich* works. It consisted in smelting down the crusts (in a low square furnace or cupola of 4 to 8 square feet sectional area and not over 8 or 9 feet high, provided with exterior crucible) together with tap or puddle cinder as iron flux to help to carry off the zinc, a large excess of slag, and sometimes lead matte.\* The blast required is of considerable volume, but low pressure (under  $\frac{1}{2}$  inch mercury). Any copper in the crust is taken up by the matte, while the silver goes into the lead. None of the zinc, however, is recovered, and the losses of lead and silver in flue-dust and foul pasty slags are considerable ; the process has, therefore, been superseded by the distillation process hereafter described, as regards the treatment of silver crusts. At one or two places, however, it has survived up to quite recently for the treatment of gold-copper crusts poor in silver, for the bullion obtained by distilling them is very coppery ; whereas by smelting them with matte a comparatively soft and easily cupelled bullion is obtained.

2. **Cordurie's Process.**—This process, first used at Havre in 1866, is still to be found in a few English and Continental works, notably, in the former, *Walker Parkers'* Flintshire works, and, in Germany, at *Altenau* and *Lautenthal* (Harz).

The same apparatus is employed as for softening and refining the base bullion.† The charge of zinc crust is 10 tons, which, owing to its semi-oxidised condition, takes eight hours to melt down. As soon as it reaches a cherry-red heat, dry steam, at 30 to 40 lbs. pressure, is turned on and blown through for about four hours. Large quantities of zinc and lead fume (containing some silver) are given off and drawn by means of a steam jet through a suitable series of condensing pipes, which must be water sealed in order to avoid explosions, as the steam becomes partially dissociated and the escaping gases contain some oxygen as well as large quantities of hydrogen. The pots are rapidly eaten away, only lasting, as a rule, for twelve charges. The consumption of fuel is 13 per cent.—viz., 9 per cent. for heating the pots, and 4 per cent. for production of steam. The products are about equal parts by weight of rich lead and of rich oxide, which floats

\* Hofman, *op. cit.*, 1906, p. 483.

† v. Figs. 218 and 219 and accompanying description in Chapter xiv.

on the top and is skimmed off, besides flue-dust, the amount of which is not stated. Their composition is given in the following Table:—\*

TABLE L.—PRODUCTS FROM CORDURIE'S PROCESS.

	Rich Lead.		Rich Oxides.		Flue-Dust.
	Altenau.	Lautenthal.	Altenau.	Lautenthal.	Lautenthal.
Pb, . . . . .	96·3448	95·1404	37·845	30·065	0·71
PbO, . . . . .	..	..	32·14	36·87	28·20
Zn, . . . . .	0·0027	0·0023	1·35	1·90	0·12
ZnO, . . . . .	..	..	23·37	23·24	69·60
Cu, . . . . .	0·8279	0·4645	1·12	1·24	0·17
Ag, . . . . .	2·4100	3·6500	1·245	1·855	0·03
Bi, . . . . .	0·0142	0·0169	..	..	..
Bi <sub>2</sub> O <sub>3</sub> , . . . . .	..	..	0·43	0·44	..
Sb, . . . . .	0·3914	0·7201	..	..	0·17
Sb <sub>2</sub> O <sub>3</sub> , . . . . .	..	..	1·06	0·57	..
As <sub>2</sub> O <sub>3</sub> , . . . . .	..	..	tr.	..	tr.
Fe, . . . . .	0·0054	0·0044	..	..	..
Fe <sub>2</sub> O <sub>3</sub> , . . . . .	..	..	1·44	3·82	0·34
Al <sub>2</sub> O <sub>3</sub> , . . . . .	..	..	..	..	0·24
Cd, . . . . .	tr.	tr.	tr.	tr.	..
Ni, . . . . .	0·0036	0·0014	tr.	tr.	..
Insoluble, . . . . .	..	..	..	..	0·42
	100·0000	100·0000	100·000	100·000	100·00

The rich lead is cupelled direct, the flue-dust is leached with dilute sulphuric acid to dissolve out the zinc, leaving the silver in an insoluble plumbiferous residue for further treatment. The oxidised portion was formerly added, a portion at a time, to the rich lead during cupellation, by which means the greater part of its silver contents became concentrated in the lead, while the oxides removed from the furnace were subsequently smelted with lead ores to recover the remainder. By this process, however, not only was the zinc all lost, but the volatilisation loss of silver was very high. Nowadays, at Lautenthal, the zinc oxide is first removed by Schnabel's process, when the residue is very perfectly absorbed by the lead-bath in cupellation.

2a. Schnabel's process,† in brief, consists in dissolving out the zinc (and copper) oxide by means of a cold 9 per cent. solution of *ammonium carbonate*, in revolving cylinders holding 1 to 1½ tons, the operation taking twelve hours. The mass is passed through a filter press, which retains the lead and other oxides and the silver; the solution, containing only zinc and copper in solution, passes to a second revolving cylinder, in which it is treated with metallic zinc to precipitate the copper; and then to iron stills, in which it is boiled with steam. The zinc is precipitated as basic carbonate, while

\* The composition of the original crusts is given in Table XLVIII.; the present analyses are from Schnabel, *op. cit.*, vol. i., p. 562.

† *Zeitschrift für Berg- Hütten- und Salinen-Wesen*, Bd. xxviii., 1880; also Schnabel, *op. cit.*, pp. 564-571.

the ammonia and one-half the carbonic acid are expelled and condensed in a series of iron Woulff's bottles, through which carbonic acid gas from burning fuel is also passed to make up for that removed by the precipitate. The basic carbonate is converted in small reverberatory furnaces into white zinc oxide, which, however, is inferior in covering power to that produced by sublimation. The consumption of coal is 1 to  $1\frac{1}{2}$  tons per ton of oxides treated.

The advantage of the process consists rather in the decreased loss of silver\* than in the value of the zinc oxide produced, which is not sufficient to cover the cost of treatment. The process is now also in use at *Hoboken* (Antwerp), for treating the oxidised portion of the rich aluminium-zinc crusts after liquation. The residues are scorified on a lead-bath and the resulting slag does not contain over 0.06 per cent. silver (say 20 ozs. per ton), whereas the slags produced by scorifying the oxides in their original condition without previously extracting the zinc, contained at least 0.3 per cent. silver (say 100 ozs. per ton).

**3. Parkes' Distillation Process.**—This process, since the introduction of plumbago retorts by Morgan in England, and, subsequently, by Balbach in America, has become well-nigh universal for treating zinc crusts. It offers the following advantages:—(1) Quick yield of practically all the silver in the form of rich lead; (2) simple and inexpensive plant; and (3) recovery, in a metallic condition for re-use, of 60 per cent. of the zinc originally used in desilverisation.

**Furnaces Employed.**—At most refineries in England and on the Continent fixed furnaces are preferred, but in America, where saving of labour is the great desideratum, the tilting furnace of Faber du Faur, used at *Mechernich* and at one or two other places in Europe, is almost universal.

Of fixed furnaces, three principal forms may be mentioned, the first of which is adapted either to direct or to gas heating, the other two being invariably heated by flame from a separate fireplace or by gas. In all the illustrations the same lettering is employed to facilitate reference.

*a.* The *Morgan* crucible furnace was formerly in use at a majority of English refineries as well as at Freiberg. Its construction is plainly shown in Fig. 254. The plumbago crucible *R*, 22 inches high, 15 inches diameter, and 2 inches thick, holding  $5\frac{1}{2}$  gallons, is provided with a hood and pipe (3 inches diameter) of the same material, to the latter of which is adapted a sheet-iron condenser, *c*. The fireplace, *b*, is packed with coke all round the crucible, which rests on a firebrick pillar, and the top of the furnace is removed by a chain and tackle, as required, while the products of com-

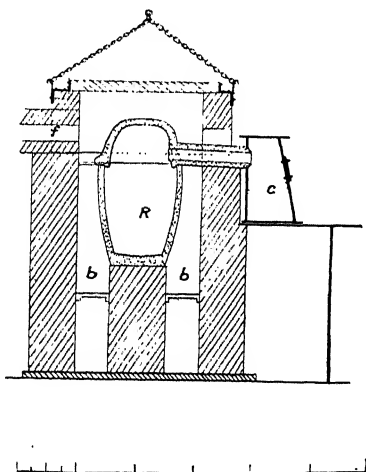


Fig. 254.—Morgan Crucible Furnace.

\* The silver extracted is usually  $1\frac{1}{2}$  to 2 per cent. above the contents of the oxides as shown by ordinary assay.



bustion escape by the flue *f*. The charge is about 5 cwts. of crust, taking eight hours to work off, and the crucible lasts for forty to eighty charges. The zinc is removed from the condenser in the form of a lump. The rich lead is dipped out by means of a small pot-shaped dipper attached to a long handle. Other particulars are given in Table LI.

*b.* The Continental *tube-retort* furnace, shown in Figs. 255 and 256,\* in which *R R* are the retorts, *c* a condenser, *b* the gas producer, and *o o* channels underneath the hearth of the furnace, through which heated air is admitted to the heating or combustion chamber. Furnaces of this description are in use at *Ems*, *Braubach*, *Tarnowitz*, and other places in Germany, also at *Hoboken* (Belgium). The retorts may be of plumbago, or, more simply, of mixtures of clay and coke, which, with care in use, last almost equally well. As many as five retorts are placed side by side across the hearth of a reverberatory. Retorts have been made 2 feet in diameter to hold as much as 1 ton each, but the best results are given by small retorts holding about 5 cwts. each. The condensers are of fireclay and plumbago, and the rich lead, after the retorting is completed, is tapped from the bottom of the inclined retort into small moulds. The distillation lasts twelve hours, and each retort lasts from 60 to 130 charges. Other particulars are given in Table LI.

*c.* The *regenerative gas-fired fixed bottle-retort* furnace in use at Port Pirie (which has replaced the non-regenerative furnace formerly in use and figured in the first edition of this work) is shown in Figs. 257 and 258.† These furnaces are built in blocks of two back to back, each holding two retorts, or four retorts to a block, each block being served by a gas producer shown in section in Fig. 257. The producers are supplied with air blown in by a steam jet under the grates, so that they make a mixture of water gas and ordinary producer gas, and they have convenient poke holes for clinkering. The current of gases is reversed each half-hour, not automatically, but by means of a bell signal given by an ordinary clock. Two men look after three blocks of twelve furnaces, and the full consumption is only 15 per cent. by weight. The retorts employed are shown in Fig. 259. They last on an average from 50 to 60 charges, and on account of their heavy cost are always used until they break. The space underneath each retort is lined with cast-iron plates and provided with a taphole, in order to facilitate cleaning up the contents of a broken retort. The condensers used with these retorts are cylindrical in shape, made of sheet iron lined with a mixture of cement and limestone 2 inches thick, and provided with a taphole. They are carried on angle-iron trestle frames resting on 4-wheeled trucks.

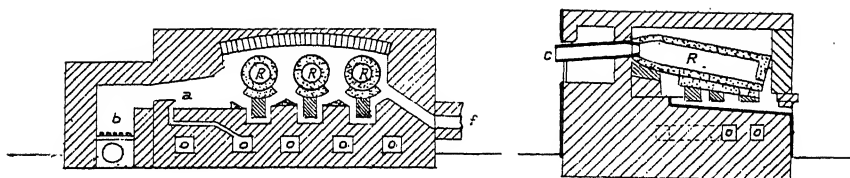
In all these fixed furnaces the rich lead has to be dipped out by means of long wrought-iron dippers made of a piece of 4-inch pipe on a long handle, or else tapped out from the bottom of the retort; in the tilting furnace the finished charge is poured out direct into moulds.

*d.* The *Faber du Faur tilting* furnace, used throughout the United States, and at a few European localities, is shown in Figs. 260 to 263.‡ It consists of a cubical box of firebrick set in a cast-iron frame swung on trunnions and adjustable by means of a hand wheel working a worm gearing, which.

\* Schnabel, *op. cit.*, p. 555.

† Delprat, *Trans. Aust. I.M.E.*, vol. xii, 1907; and Bayly, *ibid.*

‡ Egleston *Metallurgy of Silver, Gold, and Mercury in the U.S.*, vol. i, p. 104.



Figs. 255 and 256.—Tube-Retort Furnace.

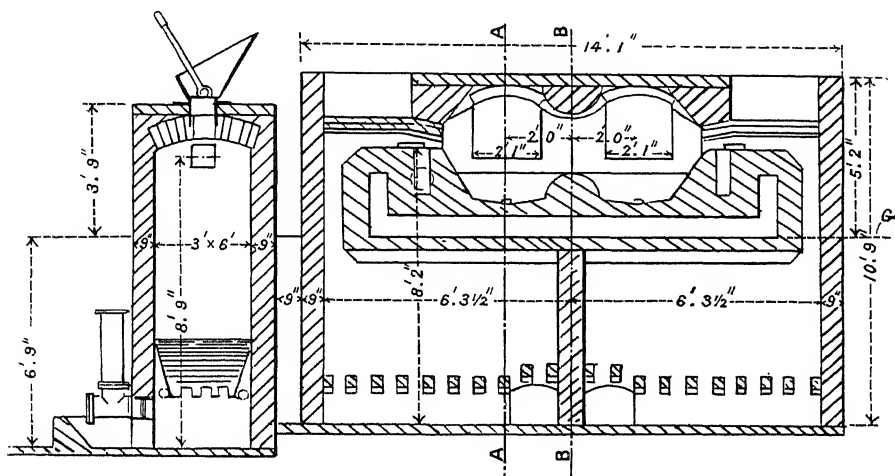
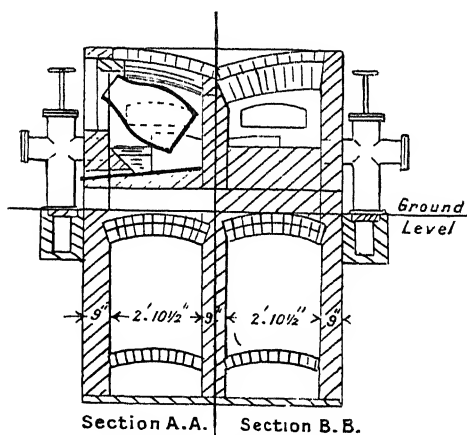


Fig. 257.—Sectional Elevation—Regenerative Gas-fired Retort Furnace (Port Pirie).



Section A.A. Section B.B.

Fig. 258.—Section—Regenerative Gas-fired Retort Furnace (Port Pirie).

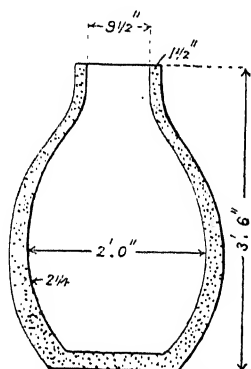


Fig. 259.—Bottle-shaped Retort (Port Pirie).

however, in view of the light weight of the furnace, is generally replaced by a cross-arm welded on to the extension of the handle, whereby it is quite easy to revolve the furnace and adjust its tilt for pouring. There is an opening in front for the neck of the retort, one at the back for the products of combustion, and another at the top for charging coke when direct-firing is employed.

The retorts are of plumbago and hold from 800 to 1,000 lbs., the usual size being 36 inches to 38 inches long, 8 inches wide at the neck, 18 inches to 22 inches at the belly, and 13 inches to 16 inches at the bottom; the

Fig. 260.

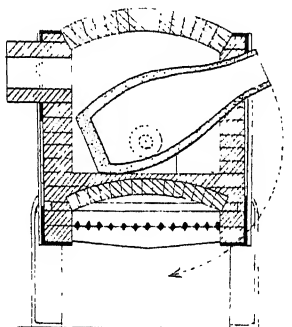


Fig. 261.

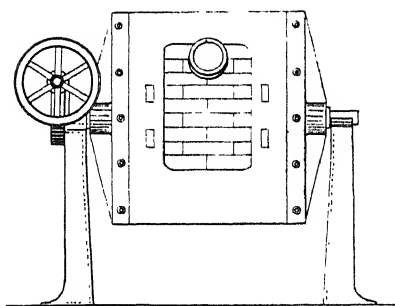
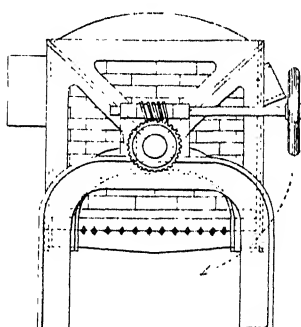


Fig. 262.

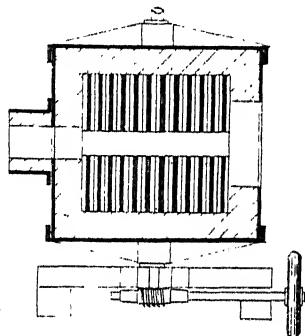


Fig. 263.



Figs. 260 to 263.—Faber du Faur Tilting Furnace.

thickness varies from  $1\frac{1}{2}$  inches at the sides to 2 inches at the bottom. Condensers adapted to the furnace are of various forms and materials, such as cast iron, plumbago, and sheet iron lined with brick or clay; they may rest upon a tripod, hang from hooks on the furnace frame, or rest upon a wheeled truck so as to be more easily handled. They may be closed entirely, or, as is more usual, have an aperture for escape of the CO and other uncondensable fumes, and the condensed zinc is usually tapped at intervals into small moulds. Probably the commonest type of condenser

is made simply of sheet iron  $\frac{1}{8}$  inch thick bent to form a pipe some 10 inches in diameter by 18 inches long, closed at the end save for a 1-inch hole, lined with from 1 to 2 inches of fireclay, and fastened to the neck

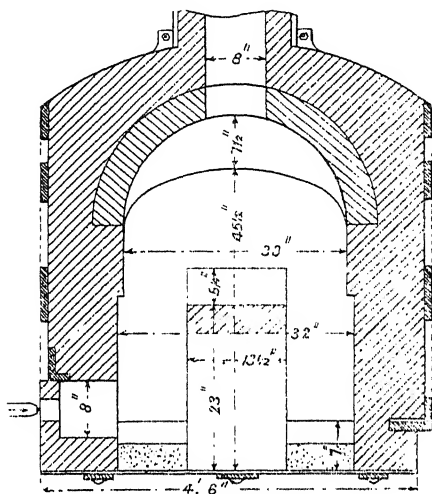


Fig. 264.

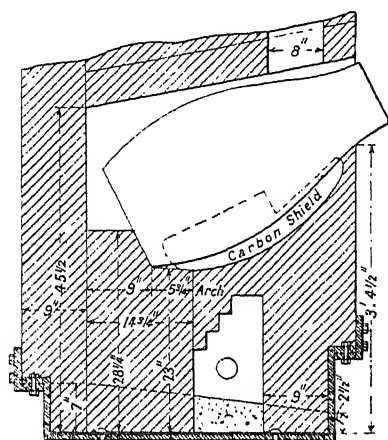


Fig. 265.

Figs. 264 and 265.—Oil-burning Tilting Furnace.

of the retort by a couple of light chains attached to the side of the furnace in such fashion as to hang slightly downwards.

Oil-firing has come into use of late years, and is a great improvement. The construction of oil-fired tilting furnaces, as used at *Perth Amboy*, is shown in Figs. 264 and 265,\* while Fig. 266 is a section of the oil burner employed. The carbon shield shown in Fig. 265 is a piece of an old retort which is used to protect the working retort from the direct impact of the intensely hot oil flame.

**Mode of Working.**—Whatever the type of furnace the operation of distilling is substantially the same. The zinc crust, broken to pieces the size of a nut, or smaller, is mixed with 1 or 2 per cent. of powdered charcoal, and charged into the already heated retort by means of a scoop. The temperature is then raised to a bright yellow heat, and more of the crust added as the mass begins to melt down, until, after a half to three-quarters of an hour, the retort has received its full charge. The condenser, which is usually of sheet iron, with a small hole for the escape

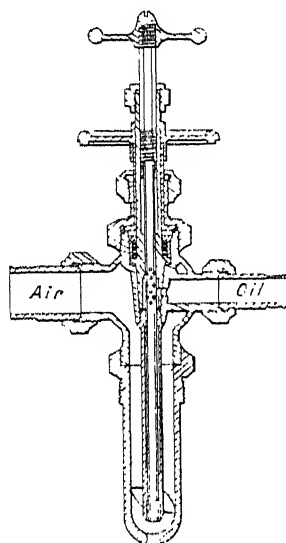


Fig. 266.—Oil Burner.

\* *E. and M. J.*, Jan. 12, 1907, p. 84.

of vapours, is then luted on, and the temperature kept as high and as steady as possible, for any irregularity causes accumulations of crust on the lead, and of blue powder on the neck of the condenser. Zinc commences to distil over when the temperature reaches  $670^{\circ}\text{C.}$ , and 90 per cent. of it has come over by the time the temperature has risen to  $1,200^{\circ}\text{C.}$ , but it is not completely expelled before a temperature of  $1,290^{\circ}$  to  $1,300^{\circ}\text{C.}$  is reached. The condensed zinc is drawn off at intervals by opening the taphole and collected in moulds; more of it being always obtained from silver- than from gold-crusts, which, indeed, in some cases, as at Port Pirie, yield no zinc at all. The distillation takes from six to twelve hours, according to the size of the retort and the type of furnace employed, and the end is known by observing that no more zinc continues to be condensed after tapping off that in the condenser. The lead is then ladled out through the neck of the retort in furnaces belonging to classes *a* and *c*, tapped out from the bottom in those of class *b*, or, finally, poured by tilting, in the case of the Faber du Faur furnace. In any case, after emptying the retort of lead, it is essential that all retort dross, charcoal, and slag adherent to the sides of the retort be carefully scraped out through the neck with an iron scraper, as also, in the case of direct heated retorts, that clinkers adherent to the outside be removed. The latter operation is much facilitated by coating the outside of the retort with a thick clay wash. A quantity of fine charcoal is then thrown into the retort to prevent formation of litharge, which would soon cut through the sides, and then another charge is added. With a good system the time lost in charging and refilling should not exceed twenty minutes.

Inclined retorts are frequently turned round in their supports in order to expose a different part of the retort to contact with the surface of the lead-bath; they are best rotated a quarter of a turn after each ten or a dozen charges.

Rösing \* proposed to facilitate distillation in a tilting furnace by pouring in molten pig iron to expel the zinc. After all the zinc was condensed, the furnace was tilted, the molten iron poured off (to be used over again), and then the lead poured out as usual. The innovation, however, has not stood the test of practical convenience in working.

**Products of Distillation.**—The products are:—(1) Condensed zinc, (2) retort bullion, (3) retort dross, and (4) blue powder.

The zinc distilled off is comparatively free from precious metals, as is illustrated by the fact that zinc obtained by retorting rich cyanide precipitates containing 40 per cent. of precious metals contains only 7 ozs. of silver per ton. Such loss as does take place is rather mechanical than due to volatilisation, at all events so long as the temperature does not rise above  $1,500^{\circ}\text{C.}$  As the condensed zinc is all used over again there is no absolute loss.

The retort bullion contains the greater part of the precious metals contained in the alloy or liquated crust charged, more especially when treating silver crusts, and when the softening of the work-lead was properly performed. It is treated by cupellation, as will be seen hereafter.

The retort dross is always as rich in precious metals as the original crust; in the case of that from gold crust it is often much richer in that metal than

\* *Berg- und Hüttenmännische Zeitung*, 1890, p. 369.

the original crust, and indeed than the retort-bullion obtained at the same time. Further reference will be made on another page to this product and to the blue powder as bye-products of the process.

**Examples of the Distillation Process.**—At *Perth Amboy*,\* there are eighteen oil-fired Faber du Faur furnaces of the pattern shown in Figs. 264 and 265, each of which takes per charge worked off 50 gallons of petroleum residue. The charge is 1,200 lbs. of crust broken small, together with a little charcoal, and the operation lasts six to seven hours; the condenser runs upon a travelling support for more ready removal, and the retort lead upon tilting the furnace runs by means of a little clay-lined launder direct into one or other of the seven cupelling furnaces.

At the *National* plant† (S. Chicago) there are twelve similar oil-fired furnaces. The retort bullion is poured from the retort into a tilting pan lined with firebrick carried on a travelling truck. After wheeling to the cupelling hearth compressed air is admitted below a piston which raises the pan and allows its contents to be poured direct into the cupel hearth.

At *Peñarroya*,‡ the coppery crusts resulting from the low-grade work-lead treated are retorted in fixed crucible furnaces regeneratively gas-fired, but the coppery dross produced which contains 20 to 25 per cent. copper is larger in amount than the retort-bullion, and has to be treated by matting, as already described.§ These furnaces are very similar to those at Port Pirie next to be described, and the condensers are supported on trucks as at that place.

At *Port Pirie*|| the gas-fired retorts shown in Figs. 257 and 259 are employed. The charge for each is 12 cwts., which, however, cannot be all added at once, but is put in little by little by means of a long-handled scoop as the charge melts down, about one shovelful of charcoal being roughly mixed with it. Charging and melting down takes about three-quarters of an hour. The condenser is a cylinder of cast iron lined with 2 inches of cement and crushed limestone, provided with a taphole at its lower end, and resting upon an iron frame supported by a little four-wheeled truck, so that it is easily run into position and the joint with the retort plastered with clay. Retorting this large charge takes about twelve hours, after which the condenser is run back, the dross skimmed, a row of moulds on a trolley is run up, and the bullion is dipped out by means of a long-handled baler by the operator, who stands upon a movable platform, and is protected by gauntlets and by a sleeved apron of sacking. Skimming, dipping, and scraping take about an hour. The weights and assay values of the different products are given in Table LI.

The retort bullion from silver crusts at Port Pirie is treated by cupellation in the ordinary way (*v.* Chapter xvii.). The retort bullion from gold crusts, however, is comparatively poor in silver and even in gold (much of the gold being contained in the comparatively large quantity of dross produced). It is, therefore, allowed to accumulate until there is enough for a charge (35 tons) of the "Special Parkes' pot," in which it is melted down slowly,

\* O. Pufahl, *Z. f. B. u. H. W.*, 1905, p. 400.

† O. Pufahl, *Zeits. f. B. u. H. W.*, 1905, p. 400.

‡ *Private Notes*, 1908.

§ Together with the dross from the softening furnace; see Chapter xiv.

|| *Private Notes*, 1896; also Delprat, *T. Aust. I.M.E.*, vol. xii., 1907, pp. 1-29; and Bayly, same volume, pp. 80, *et seq.*

TABLE LI.—WORK DONE IN ZINC-CRUST DISTILLATION FURNACES.

Reference.	1	Bagilt.	Average, U.S.A.	Branbach.	Freiberg.	Port Pirie, S.A.		Dapto, N.S.W.	
						Silver Crust.	Gold Crust.	Silver Crust.	Gold Crust.
			2	3	4	5	6	7	8
Type of furnace used,	a		d	b	a	c		d	
Weight of charge, . . . lbs.,	550		1000	605	500	1344		..	
Time required, . . . hours,	8		8	12-13	8-9	16		..	
Kind of fuel, . . .	coke		coke	coal	coke	gas		coke	
Consumption of fuel, . . . %	62.5		55	60-70	90	15		91	
Life of retort, . . . charges,	40		40	150	80	80	50-60	25	
Products per 100 parts of crust—									
Retort bullion, . . . weight %	..		70-80	60	57.2	84	..	..	76.6
" silver contents, ozs.,	..		1200-3200	3500	2401	3200	170-250	..	2627
" gold contents, "	..		..	..	16.1 <sup>10</sup>	..	2-3*	..	274
Retort dross, . . . weight %	15		5-8	3	15.8	3.2	35-40	..	10.9
" silver contents, ozs.,	..		..	..	1503	2400	126	..	1740
" gold contents, "	..		..	..	3.1 <sup>10</sup>	..	3	..	240
Blue powder, . . . weight %	..		..	..	7.2	1-2	..	..	4
Condensed zinc, . . . "	15		10	14	29.5	10.5	none	..	8.7
Percentage of zinc recovered, . .	45		60	60	50-90	67	..	..	60
Composition of crust treated—									
Silver, . . . ozs. per ton,	..		..	..	1486	3000	200	1760	1650
Gold, . . . "	..		..	..	5	..	..	0.17	280
Percentage of zinc contents of crust recovered, . . . %	..		80	96	90	..	none	..	60
Rich alloy, copper contents, . . . %	..		..	..	..	..	5.5	..	..
" zinc contents, . . . %	..		..	..	..	1.3 to 3.3	0.3	..	..
Retort dross, lead contents, . . .	..		..	..	..	56	60	..	..
" copper contents, . . .	..		..	..	..	..	12	..	..

\*Always about 12 ozs. per 1,000 ozs. of silver.

References.—1, 3, and 4. Schnabel, *Handbuch der Metallhüttenkunde*, vol. i., pp. 553-557. 2. Hofman, *Metallurgy of Lead*. 5. and 6. *Private notes*, 1896; and Bayly, *T. Aust. I.M.E.*, vol. xii., 1907, pp. 80, at seq. 7 and 8. Blakemore, *Trans. Aust. I.M.E.*, vol. v., 1898, p. 248.

yielding from 7 to 9 tons of dross, which contains most of the copper and enough gold for cupellation. This special pot dross is treated together with the retort dross upon a sweat cupel. The drossed bullion is now treated for its precious metal contents by means of three zincings. The first with 700 lbs. of zinc yields a "special gold skim," which is not pressed, on account of the danger of driving back particles of it into the pot, and so losing gold, but only drained as thoroughly as possible and then retorted, yielding a "special gold dross," which is cupelled together with the gold retort and special pot drosses above mentioned, and a "special gold retort bullion," which contains gold at the rate of from 30 to 40 ozs. per 1,000 ozs. of silver, and is cupelled up to "special doré bullion."

The residual lead bullion in the pot after separation of the gold is zined twice for silver in the ordinary way.

**Comparison between the Different Furnaces.**—The Faber du Faur furnace requires less labour, but consumes more fuel than the fixed furnaces, and is always more expensive in retorts, not only when direct-fired, but even, though to a less extent, when oil-fired, the average life of the retorts being from 25 to 50 charges. Hence, where the item of labour is of greater importance than either the cost of fuel or that of replacing retorts, as in the United States, the use of this furnace is general. In Australia labour is dear, but fuel is dearer than in the U.S., and the first desideratum is economy of the costly retorts, which can only be obtained from England; the tilting furnace, consequently, is at a disadvantage compared with the large gas-fired fixed furnace in which the retorts last nearly double as long, and in which there is a great economy of fuel due to the adoption of the regenerative principle. Gas- or oil-fired furnaces have a great advantage over those with direct coke firing in case of a broken retort, the recovery of the contents of which from among a mass of half-burnt coke is a troublesome job.

#### TREATMENT OF VERY RICH ZINC-SILVER ALLOYS.

(From the Al-Zn Process.)

**4. The Sulphuric Acid Process.**—In this the alloy is melted, granulated in water, and then treated with chamber acid in lead-lined iron pots. The zinc is dissolved out as sulphate, which, after running the solution through an asbestos filter containing scrap zinc to retain traces of silver, can be crystallised out for market. The silver and lead remain together in a semi-crystalline slime, which is run through a filter press; the cakes are moistened with a saturated solution of borax, dried, and melted.

**5. The Electrolytic Process.\***—The alloy is cast direct from the liquation pots in which it has been subsequently melted, into moulds, 14 inches by 4 inches by  $\frac{1}{4}$  inch, to form the anode plates. The cathodes are circular sheets of zinc fixed upon a horizontal spindle which revolves just above the surface of the bath. The electrolyte is a solution of  $MgCl_2$  saturated with  $ZnCl_2$ ; its specific gravity is 1.2 to 1.27. The zinc is deposited in sheets with an irregular warty surface; they contain 0.0099 to 0.0044 per cent. Fe, 0.0114 to 0.0210 per cent. Cu, 0.0341 to 0.0500 per cent. Pb, tr. to 0.002 per cent. Ag, and 99.9466 to 99.9226 per cent. Zn; the product selling as c. p. zinc at a high price. The silver slime contains, on an average, 7.4 to

\* *E. and M. J.*, Dec. 2, 1893.



80 per cent. Ag, 10 to 12 per cent. Pb,  $1\frac{1}{2}$  to 5 per cent. Cu, 0.2 to 0.5 per cent. Zn, with some As, Sb, and Fe. It is stirred in a leaden vessel with very dilute sulphuric acid to dissolve metallic oxides, and pieces of iron are then added to reduce any AgCl present. The purified slime, containing 80 to 85 per cent. Ag, and about 15 per cent. Pb, is pressed into cakes, which are dried and refined upon a cupel in charges of 2 cwts. at a time, without any addition of lead, the operation taking eight hours.

At *Friedrichshütte* (Silesia) \* zinc-silver alloy is treated by this process, As made up into anodes it contains Ag 6.3 to 11.3 per cent., Cu 6 to 8 per cent., Pb 2 to 3 per cent., Ni and Co 0.5 to 1.0 per cent., Fe 0.25 per cent., Al 0.5 per cent., Zn 81.3 to 78.6 per cent. The electrolyte is zinc sulphate, the current employed  $7\frac{1}{2}$  to  $8\frac{1}{2}$  amperes per square foot of cathode area at 1.25 to 1.45 volts. The anode mud, which contains Ag 30 to 50 per cent., Cu 50 to 80 per cent., Pb 10 to 15 per cent., the remainder being Zn, is treated with sulphuric acid to dissolve out zinc and copper, and is then added to the cupel charges. The process has, however, proved expensive, more especially in view of the small amount of alloy produced by the works, the work-lead of which is very low in silver.

#### BYE-PRODUCTS OF THE PARKES PROCESS.

Besides the bye-products obtained in softening and refining, which have been already described in Chapter xv., though the latter at all events is properly a bye-product of the Parkes process, the bye-products peculiar to the Parkes process are retort dross and blue powder, both obtained in the distillation.

**Retort dross** is a lead-zinc-copper alloy rich in silver and containing more or less carbon, silica (from the retort), and other impurities. It may be disposed of by adding a little at a time to the regular cupelling charge, or to the bath of lead low in silver with which a new cupellation hearth is usually charged. Sometimes, after drossing, it is added to the charges of work-lead in the softening furnace, in order that its impurities may be taken up in the skimmings. This practice is, however, objectionable, since it results in contaminating the residual metal with copper and zinc, neither of which are at all desirable additions.

At *Port Pirie* † the retort dross from the ordinary gold crusts, mixed with the "special pot dross" and the "special gold retort dross" already described, is smelted upon a cupel hearth in charges of half a ton at a time together with "special gold retort bullion," the copper being gradually slagged away in a litharge-slag which carries about 2 dwts. of gold and 40 to 50 ozs. of silver per ton, and the product a "gold concentrate" bullion which contains 500 to 600 ozs. of gold and 17,000 ozs. of silver per ton. This gold concentrate is brought up to doré bullion on a second cupel. The retort dross from the silver crusts, which is richer in copper than the retort bullion, and about three-fourths as rich in silver, was formerly sweated on cupels, but is now ‡ sweated in a reverberatory of 10 tons' capacity like the antimony-skimmings' furnace (*q.v.*, Chapter xiv.) built in a wrought-iron pan.

\* *Zeitschr. f. Elektrochemie*, viii., 1902, March 6, p. 140; and abstract in *Min. Ind.*, xi., 1903, p. 453.

† *Private Notes*, 1896; and Delprat, *Trans. Aust. I.M.E.*, vol. xii., 1907, pp. 1-29.

‡ Bayly, *Trans. Aust. I.M.E.*, vol. xii., 1907, pp. 80, *et seq.*

The charge at first is 200 bars (5 tons) of silver-free lead. When this is melted 750 lbs. of dross is charged and roasted to drive out zinc; then 750 lbs. of litharge from the cupels is added together with 25 lbs. of coal dust for reduction. Every two hours a charge of 750 lbs. of dross is roasted and then melted down with litharge and coal dust, as above, until the furnace is full. Once a shift the furnace is skimmed, yielding 22 to 23 cwts. of slag, the so-called "dross-furnace slag," containing Pb 50 per cent., Zn 34 per cent., and Ag 300 ozs. per ton, besides a good deal of copper, which is sent to the refinery blast furnace. When the furnace is full it is skimmed clean, cooled back, and the lead is tapped into a kettle from which it is baled, as usual, into moulds. The resulting "dross furnace bullion" contains about 3,000 ozs. of silver, and is cupelled (see next Chapter).

Reference has been already made to the fact that when distilling gold crusts, particularly those obtained from imperfectly softened bullion, the retort-dross is frequently higher in silver and in gold (especially the latter) than the retort bullion obtained at the same time. Blakemore gives\* a list of assays of these products, from which it would appear that when high in copper the dross is frequently twice as rich in silver as the retort-bullion, and more than ten times as rich in gold. The advantage of a very thorough softening of the original work-lead is thus once more demonstrated, since practically all the copper not removed by softening or in the form of pot dross comes out with the gold in the first or gold crust, and the more the copper contents of this the greater the amount of retort dross made and the higher its gold contents. The same author gives an analysis of a white hard brittle alloy occasionally found on the surface of retort bullion when treating work-lead high in gold and copper, and which contained Cu 29 per cent., Zn 55·8 per cent., Pb 3 per cent., gold 2·64 per cent., silver 9·85 per cent.†

The following figures give the composition of the products made when smelting rich liquated retort dross of the composition mentioned in a reverberatory furnace together with galena, in order to yield work-lead and copper matte:—

	Liquated Dross.	Work-lead.	Matte.	Speiss.	Slag.
Pb, per cent., . .	47·1	90·6	15·8	26·6	19·8
Cu, per cent., . .	32·0	5·4	62·3	47·0	7·8
Ag, ozs. per ton, . .	293·1	355·3	164·3	422·0	14·0
Au, ozs. per ton, . .	10·5	8·7	0·56	25·8	0·04

Here, again, we have renewed evidence of the concentration of gold in speiss, referred to in Chapter x.

At Peñarroya ‡ the very coppery retort-dross is smelted with the litharge

\* *Trans. Aust. I.M.E.*, vol. v., 1898, p. 248.

† A somewhat similar alloy, though containing much less of the precious metals, is mentioned by Hofman, *op. cit.*, 1906, p. 448, as having been found floating on the surface of ordinary gold crust. A peculiar silver-white alloy in the Author's collection, from the hearth of a fixed furnace in which the contents of a burst retort had leaked down among the bricks, contained silver 25·4 per cent., copper 14 per cent., lead 6·2 per cent., zinc 47·7 per cent., and iron 0·6 per cent.

‡ *Private Notes*, 1908.

from the cupels and the softening furnace dross in the reverberatory furnaces already referred to, the products being work-lead and copper matte. A little galena has to be added if (as generally happens) the softening furnace dross does not carry enough sulphur to carry all the copper into a matte containing from 25 to 50 per cent. of that metal, which is sold in that form. Occasionally, in addition to the copper matte, a nickeliferous speiss is produced. Analysis of one specimen of such a speiss gave copper 34.28 per cent., nickel 33.71 per cent., arsenic 24.39 per cent., lead 5.95 per cent., silver 0.70 per cent., antimony 1.62 per cent.

**Blue powder** contains a good deal of zinc oxide as well as from 4 to 18 ozs. of silver per ton. At some works it is mixed with charcoal and added to the next retort charge; at others it is placed in the desilverising pot before tapping in the softened lead, and in this case it serves to saturate the lead with zinc, thus saving part of the first zining, at the expense, however, of yielding an impure, much oxidised, and very dirty first crust. The *Omaha Works*, and perhaps some others in America, find it more convenient to ship their blue powder to European zinc works for treatment. At *Port Pirie* it is simply sent to the smelter, with other refinery products, in order to save its silver contents, no attempt being made to utilise the zinc.

With thoroughly dry and comparatively unoxidised crusts, such as are produced by the aid of the Howard Stirrer and Press, much less blue powder is produced in retorting.

**Kettle dross** goes back to the softening furnace; old retorts to the refinery blast furnace.

**Costs and Losses.**—It is difficult to arrive at the cost of treatment by any of the processes, since refineries have a great objection to publishing such figures. According to Hofman \* the average cost, including softening and refining, but exclusive of salaries and realisation charges, is from £1, 1s. to £1, 5s. per ton of base bullion; including these, the cost may be £1, 13s. to £1, 17s. 6d. per ton; while, if the loss in metals and other incidental expenses be added, the total absolute cost may amount to from £2 to £2, 10s. per ton refined. These figures, of course, refer to American works; English refineries are run much more cheaply, but it is impossible to get figures for publication.

According to Iles,† the refining costs at the *Globe Works* (Denver) were, in 1893, as follows:—

	\$	\$	s. d.	s. d.
Labour, . . . . .	1,968		8	2.40
Zinc, . . . . .	0,861		3	7.05
Coal, . . . . .	0,406		2	0.80
Coke, . . . . .	0,521		2	2.05
Supplies and repairs, . . . . .	0,289		1	2.45
Direct operating cost, . . . . .		4,135		17 2.75
The indirect cost being:—				
Interest, . . . . .	1,317		5	5.85
General exps., . . . . .	1,085		4	6.25
Parting, . . . . .	2,121		8	10.05
Bye-products (treatment of), . . . . .	1,492		6	2.60
Total indirect cost, . . . . .		6,015		25 0.75
Total cost, . . . . .		\$10,150		£2 2 3.50

\* *Op. cit.*, 1906, p. 504.

† *E. and M. J.*, Aug. 18, 1900, vol. lxx., p. 186.

The above figures refer to the period when hand-stirring was in vogue ; since the introduction of the Howard Press and Stirrer the operating costs have been reduced to \$3.45 (14s. 4.5d.), a direct saving due to these appliances of 68.5 cents, or 2s. 10½d. per ton.

The loss of lead is rarely more than 0.5 per cent. ; that of gold (in silver not submitted to parting) may be as much as 3 or 4 per cent. where the amount is small, say only a few dwts. per ton ; but with richer bullion it rarely exceeds 1 or 2 per cent. The production of silver is almost invariably in excess of the amount paid for and determined in the bass bullion by ordinary commercial assay ; the excess varies from ¼ per cent. up to 1 per cent. or even more in extreme cases.

**Comparison between the Parkes and Pattinson Processes.**—The Pattinson possesses over the Parkes process the advantages of less complexity, of a simpler plant, and, in the case of lead low in silver and already fairly pure produced by reverberatory or hearth smelting, readily yields a pure lead of specially high quality suitable for sulphuric acid chambers without any preliminary softening or subsequent refining. It is, however, not well suited to the treatment of impure blast furnace lead rich in silver, nor to gold-bearing leads.

The advantages of the Parkes process are :—

1. Lower cost of treatment to the extent of 20 to 50 per cent.
2. Production of a market lead with only 4 to 6 dwts. of silver, instead of 9 to 15 dwts. by the Pattinson process.
3. Complete recovery of traces of gold.
4. Much purer market-lead from work-lead originally cupriferous and impure.
5. Production of a lead for cupellation with 2,000 to 5,000 ozs. silver per ton, as against 500 to 650 ozs. by the Pattinson process.
6. At least 50 per cent. less loss of lead and silver.

These advantages become of greater importance the richer in silver is the work-lead treated.

## CHAPTER XVII.

### CUPELLATION AND REFINING.

**General Description.** — By whatever process obtained, the rich lead containing from 250 to 600 ozs. of silver per ton (Pattinson process), or 2,000 to 5,000 ozs. per ton (Parkes' process) has to be cupelled in order to separate its silver contents. Cupellation consists essentially in melting the lead in a reverberatory furnace, and exposing it to a blast of air, whereby the lead is oxidised together with the other base metals present, forming litharge of various degrees of purity, which runs off, while the silver remains on the hearth. Oxidisation takes place partly by direct contact with air and partly by the action of molten litharge, which absorbs an excess of oxygen, and so exercises a powerful oxidising influence on the underlying lead-bath.

As in softening base bullion, the bulk of the impurities oxidise before the mass of the lead, and can be removed by skimming; the *copper*, *nickel*, and *cobalt* in a dross (*abzug*), which forms during the melting down; the *zinc*, *antimony*, *arsenic*, and *tin*, if present, in skimmings (*abstrich*), which are, in the German process, removed after the dross and before the clean litharge, though in the English process no attempt is made to separate them from the bulk of the litharge. Such portion of the *copper* as may not have been removed by drossing is only oxidised gradually by virtue of the enormous excess of litharge, for it is noteworthy that cuprous oxide is reduced by excess of lead (*Berthier*). This oxidising power of  $\text{Cu}_2\text{O}$  comes into play to a considerable extent in ordinary cupellation, for when  $\frac{3}{4}$  per cent. of copper is present the operation is much accelerated, and the loss of silver reduced, as is noted by Kerl.\* *Bismuth* has less affinity for oxygen than lead, and remains, therefore, with the silver until almost the close of the operation, finally oxidising and colouring both hearth and litharge green; advantage is taken of this fact at Freiberg as a means of extracting and utilising traces of that metal. *Gold*, of course, accompanies the silver as it does throughout the treatment of lead ores, and, being less oxidisable, is even more perfectly concentrated.

A portion of the *silver* is oxidised and enters the litharge in that condition, but during the early part of the cupellation it is mostly reduced again by the excess of metallic lead; the first formed litharge is, therefore, much poorer than that produced towards the end of the operation. The latter probably contains silver, both as oxide and metal, according to the experiments of Wait,† who found 20 per cent. of the silver contents of a very rich bismuth litharge to be soluble in dilute acetic acid. The portion insoluble

\* Quoted by Hofman, *Metallurgy of Lead*, 1906, p. 506.

† *Trans. A.I.M.E.*, vol. xv., p. 463.

in acetic acid contained 2·7 per cent. lead as well as the remainder of the silver (2·35 per cent.), so that it is probable that the bulk of the silver in litharge exists in the condition of infinitesimal particles of a rich alloy mechanically suspended in the molten litharge. Yet another portion of the silver is volatilised together with the lead, and must be condensed in appropriate fume chambers.

Leaving out of consideration here the primitive cupellation hearths sometimes conveniently adopted in inaccessible districts,\* the cupellation process may be studied under the heads of the two prevailing types of furnace—namely, the German and the English. In the former the hearth is fixed and the roof movable; in the latter the conditions are reversed. In the German furnace the whole charge is added at once and opportunity is afforded for the successive and separate removal of cupriferos dross, antimonial skimmings, and poor and rich litharges; on the other hand, the final yield of silver is small in proportion to the size of the hearth.

In the English furnace the charging is continuous throughout the whole of the concentration, but all the impurities are removed together in litharge, which has to be resmelted. The advantages, however, of much greater capacity for a given size of furnace and smaller cost of labour and repairs, have enabled the English furnace to almost entirely replace the German type everywhere outside of Germany itself.

With both types of furnace it is advantageous to stop the process before its conclusion, and to complete it on a separate hearth; but this is very much more readily accomplished with the English than with the German type of furnace.

#### GERMAN CUPELLING FURNACES.

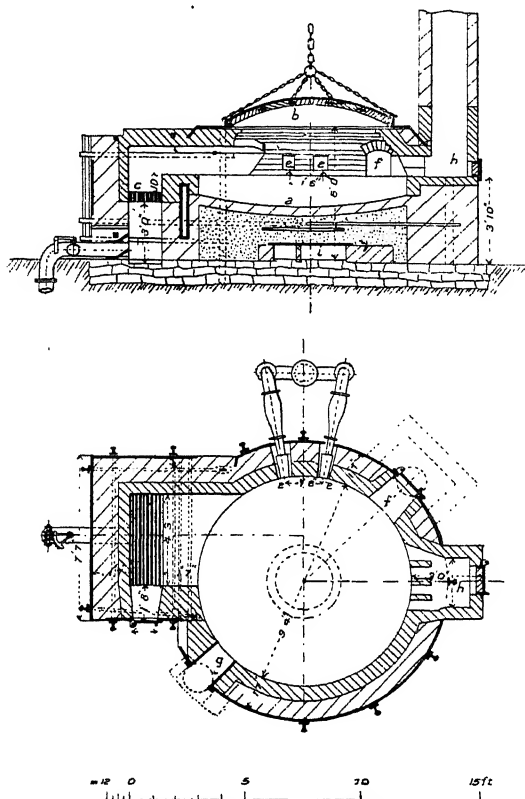
German cupelling furnaces are characterised by having a fixed hearth, the lower portion of which forms an integral portion of the furnace masonry, while the upper, or working, hearth is composed of natural marl, or a mixture of limestone or dolomite and clay (3 or 4 to 1), to which wood ashes are sometimes added (Puertollano, Spain), well tamped in against the side walls and formed into a shallow saucer-shaped cavity. The roof is a similar shallow saucer, but inverted and usually composed of fireclay tiles made to fit the curve, joined by iron cramps or rivets and bound together by a light wrought-iron framework; ordinary firebrick is, however, often used for the roofs of small-sized hearths. The iron framework is attached to chains, and can be raised and shifted by means of a crane or of a "traveller."

Two varieties of the German cupelling furnace may be mentioned—viz., the circular and the rectangular. As an example of the former (used also among other places at *Altenau*, Harz; *Hoboken*, Belgium; and *Schemnitz*, Hungary), may be described the *Lautenthal* furnace; as examples of the latter, the new *Freiberg* furnace designed by Plattner, and that used at *Przibram*.†

\* v. papers by Austin and Pfordte, *Trans. A.I.M.E.*, vol. xii., p. 41; vol. xiii., p. 185; and vol. xxi.; also Author, *Trans. I.M.M.*, vol. xii., p. 421.

† Figured and described by Hofman, *op. cit.*, 1906, p. 509.

1. The Lautenthal furnace is shown in Figs. 267 and 268 \* in which *a* is the lower hearth of red brick, upon which the working hearth of natural marl containing 66 per cent. of  $\text{CaCO}_3$  is tamped. The marl is first crushed and sifted, and then moistened sufficiently to ball in the hand. It must not be too wet, or it will not be possible to beat it to the required degree of firmness, nor too dry, or it will peel off on heating; if beaten too hard, it will crack on heating and not absorb enough litharge to prevent it from crushing; if it is too soft, on the other hand, it will absorb too much litharge. The saucer-shaped and movable roof of fireclay tiles clamped together is



Figs. 267 and 268.—Lautenthal Furnace.

shown at *b*, resting on an iron ring and hung on chains; at *c* is the grate, supplied by a gentle and easily regulated blast of air, so as to give complete combustion of the inferior slack used as fuel; it is separated from the hearth by the air-cooled firebridge. At *ee* are the tuyeres for oxidation of the lead; *f* is a working and charging door; *g* the litharge notch through which the molten litharge runs continuously; *h* is the flue leading to the fume condensing chambers; and *i*, a vault beneath the hearth to assist in keeping

\* From Schnabel, *op. cit.*, vol. i., p. 584.

it cool. The workmen at the doors *f* and *g* are protected from the lead fumes by means of sheet-iron ventilating hoods, which receive their draught from the main flue.

The charge in this furnace, with its 65 square feet of area, is 10 to 12 tons of rich lead; this is worked off in thirty-six hours direct to crude silver, which is subsequently refined in crucibles.

2. The Freiberg furnace\* is oblong in shape, though the actual hearth is elliptical inside. The roof of this furnace is flat, instead of being dome-shaped, and it is not swung on one side by a crane, but lifted by means of chain blocks, and shifted longitudinally by means of an overhead traveller. A further peculiarity in this furnace is that the litharge notch is situated at the opposite end to the firebridge, and immediately under the flue which carries away the products of combustion as well as the lead fumes.

**Mode of Working.**—A new hearth must be very carefully warmed up to prevent cracking, after which the pigs of rich lead are charged in through the working doors (frequently upon a layer of straw to protect the bottom), the roof lowered into place and luted, the litharge door closed (leaving only the litharge channel or notch), the fire urged, and the blast let on gently. The lead melts down, and in doing so leaves a dross which rises to the surface of the bath and is skimmed off. Next, a lot of brown skimmings, containing arsenic and antimony, form and run off through the litharge notch, and, finally, as oxidation proceeds, nothing but clear yellow litharge is formed. The level of the litharge notch must be regulated by cutting away the rim of cooled litharge, as it forms, with an iron hook; but only sufficiently so as to leave a rim of molten litharge from 12 inches to 20 inches wide round the edge of the lead-bath; otherwise, there is a great loss of lead and silver by volatilisation. Litharge, melting at  $954^{\circ}\text{C.}$ , requires a temperature of nearly  $1,000^{\circ}$  to be kept up in the furnace, and at this temperature the lead volatilises very fast, if too much uncovered.

The blast pressure is usually about 8 ozs. per square inch, and the volume required varies from 200 to 300 cubic feet per minute, more being required the larger the furnace, and more always toward the end of the cupellation than at the beginning.

The litharge issuing from the furnace collects either in thin sheets on a cast-iron plate, when it remains yellow on cooling; or it is collected in moulds, forming large masses which, when slowly cooled, acquire a red colour. Only the litharge produced about the middle of the process (about one-third of the whole), is sufficiently pure to be sold as such, the first third being too impure and the last too rich in silver. At the end of the operation thin films of litharge, driven by the blast over the surface of the silver, give rise to a play of iridescent colours; and, finally, the phenomenon of "brightening" takes place, as with silver assay beads.

After the brightening, two pieces of flat bar iron are stuck into the cake to serve as handles, and the hearth is cooled by water from a hose. The cake is then removed, and the hearth, after cooling, is broken up with a pick, screened, and re-tamped, only those portions which are saturated with litharge going to the blast furnace. The silver cake, still containing from 2 to 10 per cent. of lead and other impurities, may be refined on another small cupelling hearth, or in crucibles, as subsequently described.

\* For figure, v. Schnabel, *op. cit.*, vol. i., p. 588.



The time required, consumption of fuel, losses of lead and silver, &c. are for several leading German works given in the following table:—

TABLE LII.—GERMAN CUPELLATION.\*

	Freiberg.	Clausthal.	Przibram.	Müsen.
Capacity of hearth at once, . . . tons,	..	10	25	25½
Total quantity worked off, . . . "	50	10	25	25½
Value in silver, . . . ozs. per ton,	650	115	170	..
Number of tuyeres to furnace, . . .	2	2	3	1
Time taken for process—				
Preparing hearth and charging, . . . hours,	..	2	18	3
Melting down, . . . . . "	..	2	16	24-27
Removing drosses, . . . . . "	..	..	6	9
" skimmings, . . . . . "	..	4	9	144
Running market litharge, . . . . . "	..	..	23	117
" rich litharge, . . . . . "	..	28	18	..
Total time required, . . . . . "	178	36	80	300
Kind of fuel employed, . . . . .	lignite	coal	coal	wood
Fuel consumption, . . . . . per cent.,	27	21	19½	50
Weight of crude silver obtained, . . . lbs.,	..	37½	..	154
Percentage weight of dross and skimmings,	..	5	..	28
" " fowl litharge, . . . . .	76	86	..	52
" " market " . . . . .	12½	..	36	30
Assay of market litharge, . . . ozs., dwts.,	..	1·12	1·6	1·12
Percentage loss of silver, . . . . .	..	..	0·83	..
" " lead, . . . . .	..	..	4·33	..

**Variations in the Method.**—The above is the ordinary method of cupellation in the German hearth, but sometimes the rich lead is only cupelled down to from 50 to 80 per cent. silver in the large cupelling furnace, and is then tapped into moulds, the cupellation being finished on a smaller hearth, where also the silver is refined. In this case it is usual to add more lead in the large furnace as the first charge oxidises away, somewhat as in the English furnace, the total quantity treated at once thus frequently amounting to twice or thrice the weight charged at the beginning.

This practice is adopted at Freiberg, where, as already mentioned, the rich lead concentrated by the Pattinson process contains a notable percentage of bismuth. The rich lead, in quantities of 50 to 60 tons at a time, is concentrated up to 80 per cent. silver in the large furnace already described (which takes seven and a-half days), and then the concentrated alloy is cupelled in a smaller hearth to silver, which, after refining on the same hearth, is 998 fine; a bismuth litharge with 4 to 8 per cent. Bi is here formed, also a green bismuth hearth under the cake of silver, which contains 6 per cent. Bi; these are treated for the manufacture of bismuth salts by solution in hydrochloric acid and precipitation of bismuth oxychloride by dilution with water.

The same system of concentration is in use at Schemnitz,† where 20 to 30 tons are charged gradually into a hearth holding only 8 tons.

\* This information is mostly condensed from Schnabel, *op. cit.*, pp. 595-598.

† Schnabel, *op. cit.*, p. 598.

**Absorption.**—The German cupelling furnace is much more frequently employed than the English for the operation of “absorption” (*eintränken*) in which rich argentiferous ores or residues are fed gradually into the bath of molten lead, which absorbs the greater part of their silver contents, while the other metals become oxidised, and, together with the siliceous earthy constituents, are more or less completely fluxed and carried away by the litharge or raked off the surface of the bath. This practice is common at *Altenau* and *St. Andreasberg* (Harz), where silver ores containing upwards of 2,000 ozs. of silver per ton are habitually treated by this process, as much as 5 per cent. of the weight of the lead being added. It is also in use at *Hoboken* (Antwerp), and *Lautenthal*, where retort dross, oxidised zinc crusts and other residues are fed in the proportion of even 60 to 100 per cent. of the rich lead operated upon with satisfactory results, though the duration of the process is somewhat lengthened. The scorification of the retort drosses on cupelling hearths has been already referred to on p. 428.

The use of a lead-bath on a cupellation hearth for extracting the silver from sulphide precipitates obtained by lixiviation is described in the volume on the *Metallurgy of Silver*.

**Refining.**—The crude silver obtained in the German furnace has always to be refined on a separate hearth, owing to the traces of lead, copper, nickel, bismuth, &c., which it obstinately retains. The refining may be conducted on the hearth of a small cupelling furnace as at *Freiberg* and *Przibram*, or in crucibles, as at *Lautenthal*, *Frankfort-am-Main*, and *Hoboken*. The former method calls for no description here, as the manipulations are identical with those in the English method shortly to be described.

**Use of Silver Sulphate.**—Rössler found \* that silver sulphate added to molten crude silver rapidly oxidised first the lead and then the bismuth, having, however, little or no effect upon the copper. The crude silver at *Lautenthal* and *Hoboken* contains bismuth, but very little copper, so that this method is well suited to its refining. The crucibles employed are of plumbago, hold about 700 lbs. of the crude silver (950 to 980 fine), and are heated in a wind furnace supplied with air by two blast pipes just above the grate bars. In order to protect the crucible from corrosion by the slag, a section of wrought-iron pipe, 2 or 3 inches smaller in diameter than the crucible, and 7 inches long, is coated on both sides with clay, heated, and then placed on the molten bath, into which it sinks several inches. Into the centre is placed 8 or 10 lbs. of silver sulphate in lumps of egg size (prepared by dissolving silver in sulphuric acid, evaporating to dryness, fusing, and casting in moulds), and previously warmed. The silver at once begins to boil and the sulphate melts, forming a layer on the top, which has to be stirred in with an iron rod until there is no further action. The slag is then stiffened with sand or pulverised quartz and removed with a skimmer, when another addition of sulphate is made. The total quantity required is found in practice to be twice the weight of base metal removed; thus 700 lbs. of silver (970 fine and containing 21 lbs. of base metal) require about 40 lbs. of silver sulphate, which is added 8 to 10 lbs. at a time. The test for fineness is to dissolve a portion in nitric acid and add excess of ammonia, when any turbidity (bismuth) shows that a further addition of sulphate is required. Part of the silver remains in the slag, which contains  $\text{SiO}_2$  40·7 per cent.,

\* *Berg. u. Hüttenm. Zeitung*, 1889, p. 387.

PbO 33.5 per cent., Bi<sub>2</sub>O<sub>3</sub> 6.01 per cent., Ag<sub>2</sub>O 2.05 per cent., P<sub>2</sub>O<sub>5</sub> 0.64 per cent., SO<sub>2</sub> 0.61 per cent., S 0.15 per cent., Al<sub>2</sub>O<sub>3</sub> 0.43 per cent., Cu 0.45 per cent., Sb 0.02 per cent., CaO 1.73 per cent., MgO 0.25 per cent., K<sub>2</sub>O 0.64 per cent., Na<sub>2</sub>O 0.26 per cent.; the bulk of it, however, remains with the metal, having replaced the impurities.

The great advantage of this method of refining lies in the fact that the bismuth is wholly concentrated in the last slags, the first being practically free from that metal, and the small quantity of rich slag thus produced is easier to work up for bismuth than the litharge and cupel bottoms obtained in reverberatory refining. The loss of silver is also said to be smaller than in refining on the cupel, presumably on account of the smaller surface exposed, and of the fact that it is covered with slag during the operation. The cost, both for labour and material, is, on the other hand, much greater.

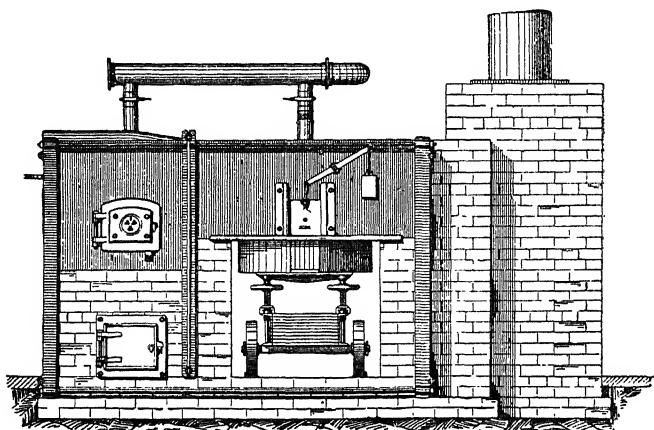


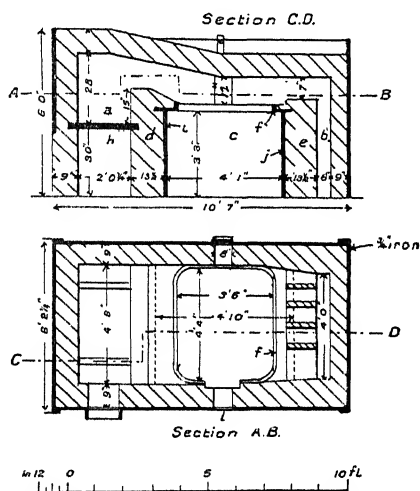
Fig. 269.—English Cupelling Furnace (Elevation).

#### ENGLISH CUPELLING FURNACES.

These are characterised by their comparatively small size, by the possession of fixed roofs and movable hearths, and by the fact that the rich lead is always charged continuously, so that the final cake of silver produced is very large in proportion to the size of the furnace.

The roof must be as low and as flat as possible, and the grate area large, since a very high temperature is required to keep such a large mass of silver in a molten condition towards the end of the process, and more particularly for refining when this is done in the same "test," as was formerly the invariable practice. The more modern practice, however, is to concentrate up to 60 or 70 per cent. silver in one furnace, and then finish the cupellation and refine in another. The temperature required for refining is so high that frequently forced draught is employed, but for concentrating natural draught is sufficient, provided the pitch of the roof is flat enough. The hearth (called a "test") may be either oval or rectangular, but the latter form is peculiar to American works.

Fig. 269 is a front elevation of an English cupelling furnace of common type, Figs. 270 and 271,\* being sections of an American furnace of somewhat similar construction but with square hearths. In the latter *a* is the fireplace, *b* the flue, *c* the vault between the firebridge, *d*, and the flue-wall, *e*, into which the test carriage runs; *f* is the iron compass ring, *h* the ashpit, *i* and *j* are cast-iron plates to support the bridge and flue-walls respectively, and *l* is the litharge door. Some furnaces have three doors at the back, but in that here figured the lead is fed, one pig at a time, through the same door in which the tuyere is inserted. The whole upper part of the furnace, as well as the front and flue are enclosed in cast-iron plates (Fig. 269), and well bound with buckstays and tie-rods (not shown). The flue exit is divided into five small flues gradually increasing in size towards the front, so as to spread the heat in that direction, where the cooling effect is greatest. In



Figs. 270 and 271.—American Cupelling Furnace (Sections).

this furnace the compass-ring is square with rounded corners (Fig. 271), the hearth having, of course, the same form and fitting tightly inside it with the help of a clay lute. It is open in front for 16 inches, in the centre of which space comes the litharge notch 4 inches wide.

Figs. 272 to 276 show details of a water-jacketed cupelling furnace described by Weinberg,\* and Figs. 286 to 292 give details of the iron plates enclosing the same furnace. The test ring and water-jacket of this furnace will be described later.

**Test Rings and Supports.**—The common style of test ring is a simple oval with a skeleton bottom of flat bars, all made of cast iron in one piece, and into which the hearth material is rammed. The pattern shown in Figs. 277 and 278 is an improvement on this for concentration work, since besides the lip or spout in front for pouring (in connection with the Lynch test support described below), the ordinary upright rim, *b*, is cast with a

\* Hofman, *Metallurgy of Lead*, 1906, p. 519.

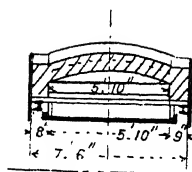


Fig. 272.—Section on A B.

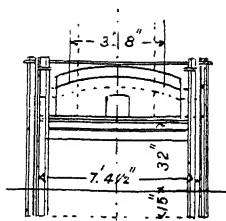


Fig. 273.—End Elevation.

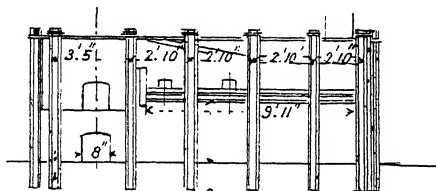


Fig. 274.—Side Elevation.

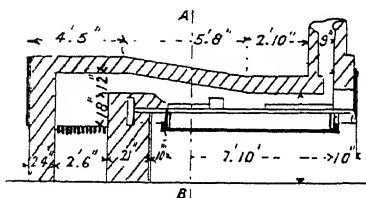


Fig. 275.—Sectional Elevation.

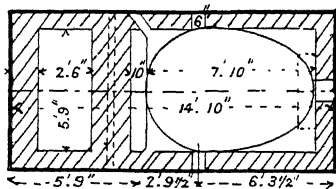
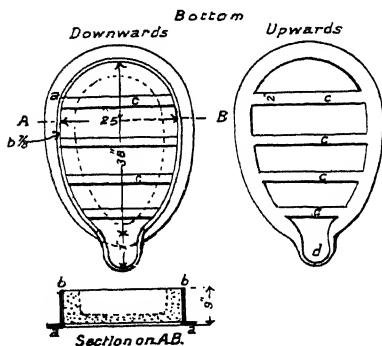


Fig. 276.—Plan.

Figs. 272 to 276.—Water-jacketed Cupelling Furnace.



Figs. 277 to 279.—Plain oval test ring with lip.

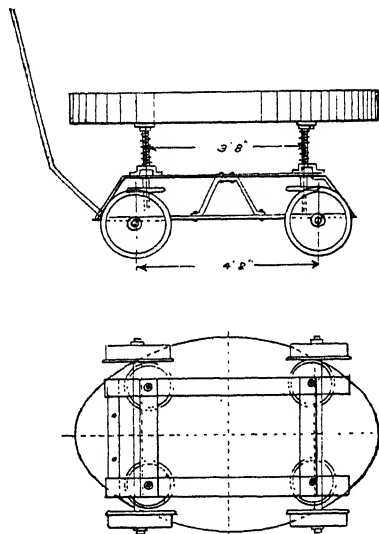


Fig. 280 and 281.—Test Carriage with oval ring.

3-inch flange, *a*, which, with its clay lute, fits against the bottom of the compass ring and so makes a closer joint, besides raising the level of the lead-bath higher than it would otherwise be, which is an advantage as regards oxidation. In both of these tests the hearth material is tamped to the required shape by the eye only.

In the oldest style of furnace the simple test ring was supported merely on transverse bars inserted across the vault, and levelled up by means of wedges. This primitive arrangement, however, gave place many years ago to the test carriage, with four adjustable jack-screws, shown in position in Fig. 269, and a somewhat different pattern of which is again shown in Figs. 280 and 281.

This form of test carriage has come into general use, and is extremely convenient, except where pouring from the test is practised, for which it possesses the disadvantage that the position of the test cannot be readily altered during cupellation, as the jack-screws when hot become very hard to turn.

Fig. 282.

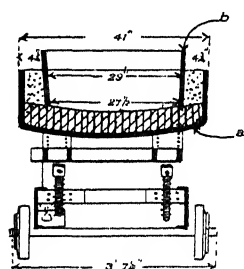
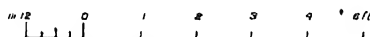
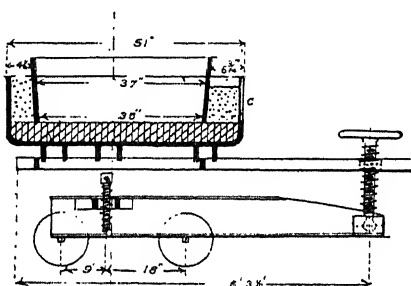


Fig. 283.



Figs. 282 and 283.—Rectangular Brick Test with Tilting Carriage.

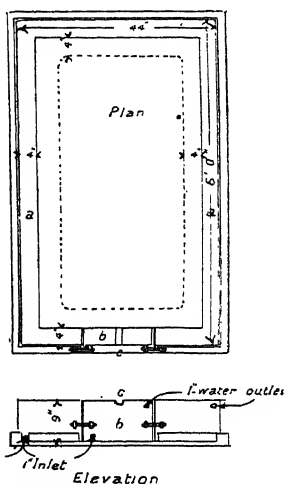
The furnace shown in Figs. 270 and 271 with rectangular compass ring has, of course, a rectangular test, one form of which, with tilting carriage, is shown in Figs. 282 and 283.

This test ring, unlike the others, has a solid cast-iron bottom, *a*, upon which is a layer of firebrick closely laid. The cast-iron pattern, *b*, shown in the figures in place, is then oiled and inserted, and the hearth mixture (cement in this case) rammed tightly into the intervening space; after which the pattern is withdrawn and the hearth dried as usual. At *c* is the aperture for the litharge notch, which consists of a slot in the front of the casting 3 inches wide, which is filled up to the required height with the mixture leaving, however, a gutter about  $1\frac{1}{2}$  inches wide by  $\frac{3}{4}$  inch deep. The advantage of a rectangular test is that, for any given width of furnace, it affords the maximum of surface, and, therefore, the maximum of oxidising effect; oval tests, however, generally stand better.

The test carriage shown with this brick-bottomed test is peculiar in being provided with a simple hand-wheel arrangement at one side of the

front for lowering the front of the test ring as the cupellation proceeds, so as to regulate the flow of litharge without cutting down the litharge notch, and also to permit of pouring the contents of the test if required.

**Water-jacketed Tests.**—With all the above test rings the litharge soon eats away the material forming the test, and, therefore, many devices have been adopted for cooling by means of water pipes coiled round the test ring, &c., combined often with a separate water breast, so as to diminish the corrosion. One of these devices in common use in America is the completely water-jacketed test of Steitz, shown in Figs. 284 and 285. The water jacket, *a*, of boiler iron is rectangular and complete, except in front, where a smaller cast-iron jacket, *b*, bearing the litharge notch, *c*, is bolted on. The only wear of the jacket is in this notch, and when it is worn through, the small breast jacket is taken out and replaced by a new one. This form of test ring for concentrating rich lead up to 60 or 70 per cent.



Figs. 284 and 285.—Water-jacketed Test.

silver has the advantages of trebling the life of the hearth material, while saving trouble with the litharge notch, which remains at a constant level and suffers no corrosion. This constant level of the notch, however, coupled with the enormous cooling effect, renders it impossible to bring up the enriched alloy to fine silver on such a test; consequently, the Steitz test is used for concentrating alone.

Not only, however, is the cooling effect with these completely water-jacketed tests so great that more fuel has to be employed to avoid retardation of the process, but the tests themselves are greatly racked by the varying degrees of intense heat to which they are subjected, and soon begin to leak at the seams. For these reasons metallurgists outside of the United States generally prefer the plain oval-flanged cast-iron test ring, with a 1-inch pipe all round, fitting tightly against the ring, and imbedded in the material of the test.

At the front is a small cast-iron tapping breast carrying the litharge notch or lip. The breast is easily replaceable when it begins to leak, though it usually lasts out from five to six tests. Before using one of these breasts over again it is tested for leaks by hydraulic pressure in order to minimise the danger of springing a leak while the cupel is running.

For concentrating retort bullion Weinberg describes \* a water-jacketed test which fits the furnace already referred to in Figs. 272 to 276, and 286 to 292, and is shown in Figs. 293 to 298. The test ring is itself made in two parts—viz., a flat base (Fig. 294), and the ring of bars which rests upon it (Fig. 293). Upon this ring stands an oval water jacket, 7 feet 6 inches in longest diameter by 5 feet 5 inches at the widest part, 9 inches high, and having a 3-inch water space, shown in Figs. 295 to 298.

Steel plate was found to be rapidly corroded if it came accidentally in contact with the litharge, even when water-cooled, and as an improvement

\* *Trans. Aust. I.M.E.*, 1901, vol. i, p. 167.

Fig. 286.

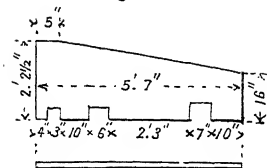


Fig. 288.

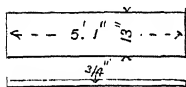


Fig. 287.

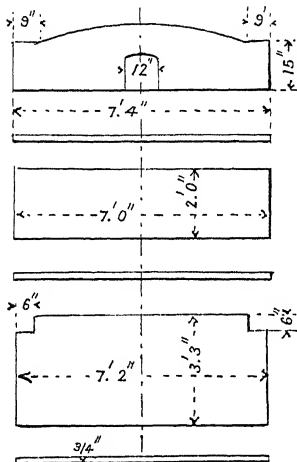


Fig. 289.



Fig. 292.

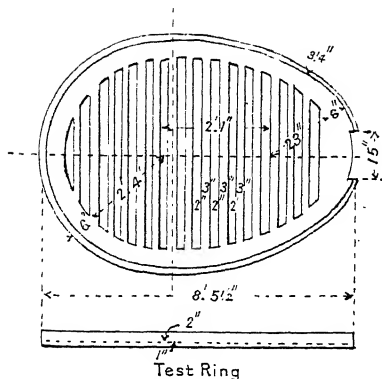


Fig. 293.

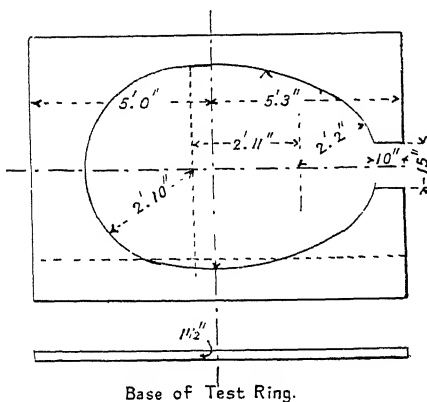
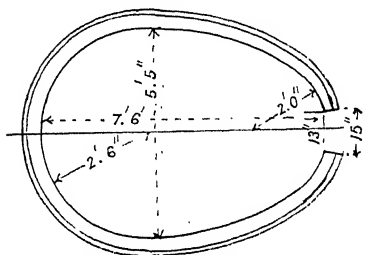
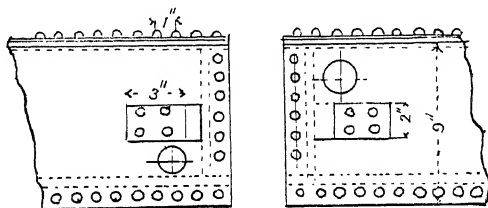


Fig. 294.

Figs. 286 to 294.—Details of Ironwork Water-jacketed Cupelling Furnace.

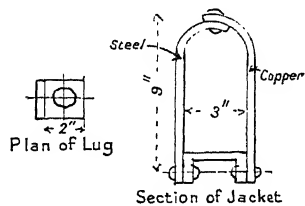


Plan of Jacket  
Fig. 295.



Ends of Jacket

Fig. 296.



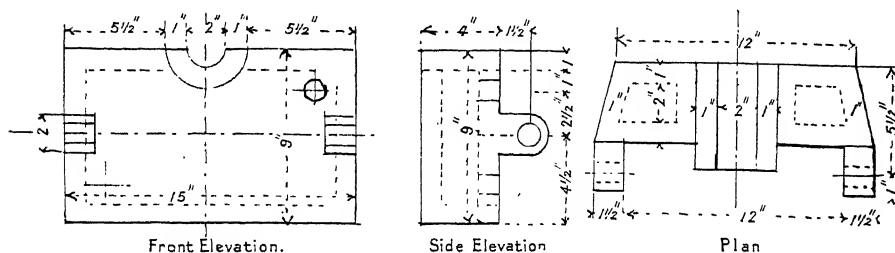
Figs. 295 to 298.—Details of Water Jacket.

### Plan of Lug

Section of Jacket



the construction shown in Fig. 298 was adopted, having a  $\frac{3}{8}$ -inch copper plate on the inside and a  $\frac{3}{8}$ -inch steel plate on the outside, rivetted on the top with copper rivets. A single jacket of this pattern lasted during no less than five years' constant concentration. The litharge breast, which fits between the ends of the water jacket is, of high-grade cast brass with water inlet and outlet of  $\frac{3}{4}$ -inch pipe; details of its construction are shown in Figs. 299, 300, and 301.



Figs. 299 to 301.—Details of Breast Water-jacketed Cupelling Furnace.

The lining of the water-jacketed test above described is 4 inches in thickness inside the jacket, and it holds, when new, 4,265 lbs., and, when worn, 4,700 lbs. of retort bullion. Its life is no less than three months when cupelling (concentrating), and two months when working up retort drosses. Upon these figures this pattern of water-jacketed test would appear to be the best appliance available for concentration, yet many large plants (*e.g.*, Port Pirie) adhere to the simple ring of 1-inch pipe all round a plain test ring. It should be remembered that the fuel consumption is always considerably greater when complete water jackets are in use.

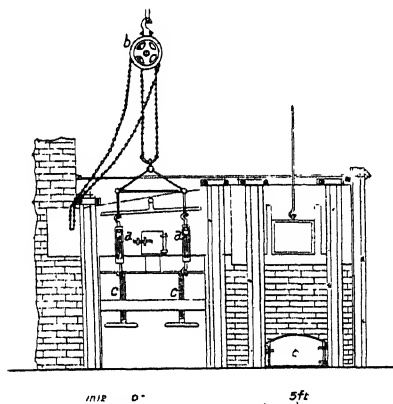


Fig. 302.—Lynch Test Support.

upon two turnbuckles, *a*, which in turn, by means of an iron triangle, hang from a differential chainblock, *b*. The back of the test rests upon jack-screws, *c*, working in a fixed bar. With the block the front of the test can be easily raised or lowered during cupellation as required, and, with the turnbuckles, either side can be raised or lowered so as to regulate the corrosion of the hearth.

**Pouring direct from the Cupel.**  
—When the system of concentrating and finishing on a separate hearth is adopted it is convenient to be able to tilt the test so as to pour the molten alloy into moulds. This can be done with the hand-wheel carriage shown in Fig. 283; but the Lynch test support, shown in Fig. 302,\* is also convenient. It consists of two bars on the lower side of the test ring, by means of which the front end of the test is supported

\* Blake, *Trans. A.I.M.E.*, vol. x., p. 220.

The support is never required for water-jacketed tests with a constant discharge level, which are, therefore, commonly mounted on one or other of the forms of carriage already figured; whereas the Lynch support is sometimes seen in America on finishing tests which are not water-jacketed, and when it is thought more convenient to pour the silver than to ladle it. The total quantity to be handled, however, on the finishing test is so small that, even in the United States, it is more common to use the fixed jack-screw test carriage and ladle out the fused silver into moulds, instead of attempting to pour it.

**Hearth Material.**—Formerly the only material used for the tests of English cupelling furnaces was *bone-ash*, which is still to some extent used in England itself for that purpose. Elsewhere, however, bone-ash has been practically superseded by mixtures of *siliceous limestone* and *clay* (in proportions of from 2 to 5 per cent. of the former to 1 of the latter), ground to pass a 12-mesh; *Portland cement*, which has given excellent results at many American works; a mixture of *cement* and ground *firebrick* has also given good results.

The following table gives a list of various mixtures which have been found to give good results. The first three are by Blakemore,\* who states that they all work "about equally well," the next four are by Piddington,† as having been found to give satisfactory results for melting down cyanide precipitates on a lead-bath, and the last is recommended by Graham‡ for similar work.

Material.	Proportions by Weight.							
	1	2	3	4	5	6	7	8
Limestone, . . . . .	7	4	8	1	4	8	4	..
Cement, . . . . .	2	1	1	2	1	1	1	2
Fireclay, . . . . .	1	1	1	1	1	1	1	..
Magnesia, . . . . .	..	4	..	..	4	..	2	..
Borate of magnesia, . . . . .	..	..	..	..	..	..	..	1
Ground firebrick, . . . . .	..	..	..	..	..	..	..	1

It will be noticed that Nos. 2 and 3 (Blakemore) are the same as 5 and 6 (Piddington).

Bone-ash tests should not be beaten down too hard, as that would make the test crack on firing. The limestone-clay type of test, however, must be tamped as hard as possible, and dried and warmed very gradually and thoroughly. In using cement it is essential to do the tamping as rapidly as possible before it commences to set. Whatever the material, it is essential that the whole quantity required should be put in almost at once and tamped continuously during the filling, for if the mixture be put in and consolidated in separate layers these are almost certain to peel.

The bottom of the test shown in Figs. 282 and 283 is made of firebrick, and the most durable tests of all are made of *magnesia brick*, which is exclu-

\* *Trans. Aust. Inst. M.E.*, 1898, vol. v., p. 221.

† *Journ. Chem. Met. and Min. Soc. of S. Africa*, 1903, vol. iv., pp. 232, 389.

‡ *Idem*, 1905, vol. v., p. 315.

sively employed at *Perth Amboy*, as well as at some other places. Most works, however, consider tests of this material to be too expensive for general use. A brick hearth is almost a necessity for bringing up argenteiferous matte to bottoms (see the volume on the *Metallurgy of Silver*).

Whatever its material, the hearth should be dried out for at least a month before commencing to warm up. A plain (not water-cooled) cement test used right through for concentrating as well as finishing may last seven days, while one used four or five hours daily for finishing only will last for months, and even a limestone-clay test, under these circumstances, lasts a month; a completely water-jacketed concentrating test of limestone-clay will last two months.

According to Weinberg,\* tests made of 5 parts cement and 1 part sand tamped rapidly over a wooden mould in a water-jacketed test of his design last three months when used for concentration.

At *Port Pirie*,† all the tests are alike, except that those used for finishing have no water-cooled lip or litharge breast. The tests measure 5 feet by 4 feet by 12 inches deep, the thickness at the top being 9 inches at the litharge end and 4 inches at the far end. The mixture employed is marble 300 lbs., ordinary limestone 120 lbs., white fireclay 60 lbs., and cement 120 lbs., or, limestone 70 per cent., cement 20 per cent., and fireclay 10 per cent. Instead of being put into the test lying bottom upwards upon a mould, the present practice is, after thoroughly mixing the ingredients dry, to bring the mixture to the consistency of very soft putty. The mixture is then, as rapidly as possible, filled into the test which lies face upwards, an oiled wooden mould the size of the cavity is then placed in position and pressed down by means of heavy weights, the space between the mould and the testing being prodded meanwhile with the fingers to get out air bubbles. When set the mould is removed, the top covered with wet bags, and the test is put aside in a warm, dry place to "season" for four to six months. The average life of tests prepared in this way is, for concentrating up to 60 per cent. concentrated bullion, three weeks, at an average of 30 bars per shift, or, say, a total of 72 tons lead passing through the test, at the end of which its capacity has become enlarged from 1 to 2 tons. On the finishing side of the cupellation house each test lasts, on an average, ten days, going constantly day and night, or while an average of 400,000 ozs. of silver is cupelled upon it.

**Blast.**—The tuyere (not shown in the Figs.) is generally a 3-inch sheet-iron pipe, which is either flattened at the end so as to deliver a sheet of air 4 inches wide by  $\frac{1}{2}$  inch thick, or else fits into a cast-iron nozzle of that size, which is preferable. The blast for cupellation was formerly always produced by means of a steam-jet injector, and this method, which allows of very exact regulation, is still in use at many places, notably at the great *Perth Amboy* plant. It is, however, more common nowadays to employ a fan or a Roots' blower. The blast pressures employed vary between 4 and 8 ozs. per square inch. At *Port Pirie*, with a flat nozzle of 4 inches by  $\frac{1}{2}$  inch, a pressure of 10 or 12 ozs. per square inch is available, but the blast is generally partly shut off, so that the actual pressure employed is from 5 to 6 ozs.

**Mode of Working.**—After gradually heating up the test to a red heat, lead is charged through the front and back openings, and, when melted

\* *Loc. cit.*

† *Private notes*, 1896, and Bayly, *Trans. Aust. I.M.E.*, vol. xii., 1907, p. 80.

down so as to fill the cavity, the blast is turned on and cupellation started. The litharge runs off at the litharge notch, and the rate of running off is regulated, partly by the rate at which pigs of lead are fed forward at the tuyere door, partly (in tests unprovided with water breasts) by the depth of the litharge notch. In any case, it is usual to keep about half the lead covered with litharge to reduce the loss by volatilisation.

The level of the lead is usually kept just below that of the notch, and litharge then overflows in waves caused by the play of the blast.

When the test has become filled with rich alloy (sometimes called "bullion concentrates"), the blast is stopped, the contents of the test removed into moulds by ladling or pouring, and a new concentration begun, the furnace working continuously as long as the test lasts.

The operation of "finishing" is conducted similarly, but at a higher temperature, in an uncooled test, until sufficient crude silver is obtained, when the addition of pigs of concentrated alloy is stopped, and the last litharge removed. The phenomenon of "brightening" is seldom or never seen on the English hearth.

Refining of the crude molten silver is sometimes conducted in a separate hearth, but generally in the finishing hearth itself, and by the blast alone. A little bone-ash, however, is sprinkled on to absorb base metal oxides, and, generally, also a few shovelfuls of soda nitre, one at a time; the slag so produced is usually not removed until just before casting. A ring of powdered brick is often made upon the surface of the silver near its edge, in order to retain within it the molten slag and nitre, and prevent them from corroding the test. Refining is concluded when the metal has a clean, smooth, brilliant surface like a mirror, which does not become tarnished on stirring, and when a small sample bar shows a finely granular silky fracture and great malleability. It is usual, however, to make careful assays before casting, in order to be sure that the bars will be above the standard for fine silver (viz., 997+).

When the bars are to be cast for market direct from the cupel without remelting, some large sticks of charcoal should be allowed to float upon the surface of the silver for ten minutes before ladling or pouring is commenced, in order to absorb some of the dissolved oxygen. Casting is occasionally done with the help of the Lynch test support already described, the silver being poured direct into the moulds, but more usually the metal is ladled out into the warmed moulds, which are oiled or filled with straw in order to prevent "spouting." The sample is best taken from each mould as poured, and just before "setting," by means of a long iron spoon, the contents of which are poured upon an inclined board resting upon the edge of a bowl of water, in which the resulting fine granulations are collected. The usual size of mould for commercial fine silver is one which holds a trifle over 1,000 ozs.

**Examples of English Cupellation.**—At *Eureka* (Nev.)\* bone-ash tests, holding 1 ton each, were filled with rich lead, containing 550 ozs. of silver, which was melted down and fresh lead added as fast as it oxidised. Once in each twenty-four hours the rich alloy was tapped, yielding six to eight bars of 60 to 70 per cent. silver. In the "finishing" operation sixty such bars were cupelled in sixteen hours, giving 16,000 ozs. of doré silver, 965 fine in silver, and 30 in gold. From the concentration some litharge

\* Curtis, *Mon. U.S. Geol. Survey*, vol. vii., 1884.

was obtained, running only 1 oz. of Ag per ton, which was reduced to market lead in a reverberatory with charcoal. The litharge from the finishing test all averaged 75 ozs. per ton, and went back to the ore furnace.

At *Perth Amboy* there are seven water-jacketed concentrating hearths, each of which works up from  $4\frac{1}{2}$  to 5 tons of retort-lead per twenty-four hours, using a steam-jet injector blast and burning 3 tons of coal per day each.

At *Port Pirie* \* there are six concentrating and two finishing hearths, all being alike in general form and dimensions, but only the former are furnished with a 1-inch water pipe round the test for cooling purposes and with a water-cooled litharge breast. In the former the bars of silver retort-bullion containing on an average 3,000 ozs. of Ag per ton are melted down with a 6-oz. blast at the rate of thirty bars (say 22 cwts.) per eight-hour shift, or 66 cwts. per twenty-four hours, producing 18 cwts. litharge per shift, the capacity of the hearth being from 1 to 2 tons, according to the amount of corrosion it has undergone. Concentration is stopped when the lead reaches 15,000 to 16,000 ozs. to the ton, and the contents of the test are then ladled out into moulds. The litharge from the concentration averages only 30 ozs. per ton, and is smelted in the refinery blast furnace together with other refinery bye-products as an addition to the ore charge.

About  $\frac{1}{2}$  ton of the bars of concentrated bullion is melted down in the finishing hearth, and cupellation is begun, more being added at the back, as usual, to supply the place of that oxidised. The rate of cupellation on these finishing tests is about 14,000 ozs. of silver per eight-hour shift, and the quantity of silver produced from one filling is about 23,000 ozs., which is ladled out by hand into moulds, sprouting and effervescing violently as it solidifies. The average fineness of the finished silver is about 997 $\frac{1}{2}$ , the chief impurity present being copper, which obstinately remains after all the lead has been removed. The litharge produced on the finishing hearth contains about 200 ozs. of Ag per ton, and goes to the softening furnaces to assist in the oxidation of antimony, as explained in Chapter xiv.

The silver is "dried," or refined, upon a perfectly new finishing test by exposure for six hours to the action of the blast, which oxidises the last traces of base metals. No fluxes are employed, but a little lime or hearth mixture is sprinkled on to the bath towards the end of the operation in order to collect the floating oxides in a stiff slag, which is then removed by touching with an iron rod, to which it adheres. After careful skimming the silver is cast into bars, which assay from 998.5 to 999.2 fine, the moulds employed being of 700 ozs. capacity and hexagonal shape, in order to avoid confusion with the shipping bars of remelted silver.

The "dried" silver being too fine for the Eastern market, for many years past the ingots have been remelted in large plumbago crucibles holding 2,200 ozs. in a wind furnace with the addition of a sufficient proportion of copper to reduce the fineness to 996 exactly. The melting is done without fluxes, and any small quantity of slag formed is removed by touching it with a cold iron rod immediately before pouring. The moulds are perfectly clean; warmed, and black-leaded, but no oil or other substance is found necessary to give a clean surface or prevent sprouting. Just before pouring, a single large stick of charcoal is laid across the surface of the metal in the crucible,

\* *Private Notes.*

and this is found to absolutely prevent all sprouting during cooling, provided the metal is poured at the right temperature—i.e., not too hot. Sometimes a little slag is found on the top of the bar, but this is so small in amount that one man can clean, chip, weigh, and stamp sixty bars in a day of eight hours. The bars average about 1,020 ozs. each.

The gold concentrate bullion obtained from the dross and retort bullion from the gold crusts at Port Pirie in the manner described in the previous chapter, and containing about 550 ozs. of gold and 17,000 ozs. of silver per ton, is treated similarly to the silver concentrate bullion on a finishing cupel in  $\frac{1}{2}$ -ton charges, yielding "doré bullion," which carries about 30 to 35 ozs. of gold per 1,000 ozs. of silver, and a litharge with 40 to 50 ozs. of silver and about 0.1 oz. gold per ton. The parting of this doré bullion is described in the volume on the *Metallurgy of Silver*.

**Losses by Volatilisation.**—There are very few figures obtainable on this point. The average loss of lead, however, on the whole refining process is supposed to vary between 3 and 8 per cent., by far the largest part of which is in the cupellation; similarly, the true loss of silver varies between  $\frac{1}{2}$  and  $1\frac{1}{2}$  per cent., mostly in the final stages. Even of these amounts, however, a portion is always recovered from the fume chambers. At *Kongsberg*,\* the loss of lead is said to have averaged formerly  $6\frac{1}{2}$  per cent. of all the lead melted on the cupel. At *Przibram*,† the cupellation losses average 0.83 per cent. of the silver, and 4.33 per cent. of the lead actually cupelled. An average figure might perhaps be 5 per cent.

Practically the whole of the volatilised silver carried away in the fumes can be collected by filtering them through bags or by the use of exhaust fans. At *Brinsdown*, where eight cupelling furnaces are at work producing a high grade litharge for the manufacture of white lead (*q.v.*, Chap. i.), the exit flue from each furnace is connected with a high-speed fan provided with jets of water entering the casing; these serve the double object of cooling the flue gases from their initial temperature of 250° to 300° C. down to 40° or 50°, and of condensing and collecting the fume, which, washed out of the fan casing by the stream of water, is recovered in settling tanks. The fumes from cupel furnaces usually contain from 30 to 40 per cent. lead; their value in silver varies according to the grade of the material treated, but may range from 60 up to 135 ozs. per ton.

In spite, however, of the volatilisation-loss of silver, owing to the use of the ordinary *uncorrected* low temperature cupellation—or combined scorification and cupellation—assay for the work-lead or base bullion treated, the working loss of silver is usually well within the assay loss, and, consequently, most refineries are able to show an apparent excess of production, or *plus* clean-up, which may reach as much as  $\frac{1}{2}$  per cent. on the total silver turned out.

**Comparison between the German and English Methods.**—The advantages of the German system of cupellation in connection with comparatively low grade rich leads, such as those from the Pattinson process, are:—

1. Smaller consumption of fuel.
2. Possibility of producing a considerable proportion of pure market litharge free from impurities and low in silver, which sometimes sells better than lead itself.

\* Percy, *Metallurgy of Silver and Gold*, vol. i., p. 512.

† Schnabel, *Handbuch der Metallhüttenkunde*, vol. i., p. 597.

TABLE LIII.—ANALYSES OF CUPELLATION PRODUCTS.

Reference.	Preliminary.			Litharge.		Silver.	
	Antimony Attending.	Litharge	Cupel Bottom.	Fine dust.	Litharge.	Modelled. Partition.	
	1	2	3	4	5	6 7	
P840.	8.22	28.374	28.899	24.41	5.134	100.3	
C240.	1.28	0.584	0.570	—	0.43	100.0	
B240.	—	—	—	—	—	—	
M240.	—	—	—	—	—	—	
A240.	—	—	—	—	—	—	
S240.	—	—	—	—	—	—	
A240.	0.04	0.014	—	—	—	100.3	
B240.	—	—	—	—	—	100.0	
A240.	—	—	—	—	—	—	
F240.	—	—	—	—	—	—	
Z240.	—	—	—	—	—	—	
N240.	—	—	—	—	—	—	
A240.	—	—	—	—	—	—	
M240.	—	—	—	—	—	—	
C240.	—	—	—	—	—	—	
S240.	—	—	—	—	—	—	
H240.	—	—	—	—	—	—	

References.—1, 2, 3, and 4. Hofman, *Metallurgy of Lead*, 1906, p. 515. 5. Wait, *Trans. A.I.M.E.*, vol. xv., p. 403. 6. E. and M. J., Oct. 28, 1905, p. 784. 7. *Min. Ind.*, vol. xv., p. 545.

The English system is specially suited to the treatment of the rich lead obtained by the Parkes process, but it possesses the following general advantages:—

1. Smaller first cost of plant for a given output of silver.
2. Saving of 50 per cent. or more of the labour cost.
3. Saving of three-fourths the cost of hearths and hearth material.
4. Smaller quantity of hearth material to be re-smelted.

Except in Germany itself, the German type of furnace has been almost entirely superseded by the English furnace with its modern improvements.

**Analysis of Cupellation Products.**—Table LIII. shows the composition of products of cupellation, chiefly at Przibram.

**Treatment of Bye-products.**—The bye-products of cupellation (leaving out of consideration special products containing bismuth, the treatment of which has been already referred to) are flue-dust, cupel bottoms, and litharge. The two former always go back to the ore blast furnace.

From the German hearths, treating lead of only moderate richness, part of the litharge is always pure and sufficiently low in silver to be sold as such, the remainder is smelted to lead, either alone or in the ore furnace. The litharge from the English furnace treating enriched lead is never poor enough in silver for market, and must be smelted, either with dry silver ores or alone, though usually a portion of it can be advantageously added to impure leads in the softening furnace, as already seen, in order to help oxidise out the antimony.

The litharge produced in concentration is, of course, always poorer than that produced in finishing; no general rule can be quoted, but litharge from concentration cupels may carry from 10 up to 60 or 80 ozs. silver, whilst that from finishing cupels may run up to several hundred ounces to the ton, as may also the litharge from cupels upon which drosses are scorified as sweated.

At *Schemnitz* (Hungary), *Sala* (Sweden), and perhaps some other places, the stream of molten litharge from the furnace runs direct into small iron cylinders filled with glowing charcoal standing on a grate over an iron pot. The litharge is completely reduced to metal by the glowing charcoal, the consumption of which does not exceed 10 per cent. by weight of the lead produced.\* Any fume given off is drawn into the main flues by means of a hood.

The flue-dust has been already referred to as containing usually from 30 to 40 per cent. lead, and from 60 to 135 ozs. of silver per ton. The cupel bottoms may contain about the same amount of lead, in each case, say, about 30 to 50 per cent., but are much richer in silver, carrying, in the case of ordinary retort lead, from 500 to 700 ozs. of silver per ton.

**Parting.**—The separation of gold from doré bullion will be fully dealt with in the volume on the *Metallurgy of Silver*, 2nd ed.

\* Theory requires less than 6 per cent.



## CHAPTER XVIII.

## ELECTROLYTIC REFINING.

THE first serious attempt to refine base bullion electrolytically was that of the *Keith* process in 1881.\* This process employed as electrolyte a solution of sulphate of lead in acetate of soda, the anodes being plates of base bullion 24 inches  $\times$  6 inches  $\frac{1}{4}$  inch thick, weighing 76 lbs., and cast by means of a revolving annular frame carrying moulds successively under the pipe delivering the molten metal from the casting furnace. These anodes, each in a muslin bag to retain the slimes, were placed upon a circular anode frame, inside which, and concentric with it, was a cathode frame, carrying the cathode plates at a distance of 1 inch from the anodes. The current used was about 1,000 amperes at 2½ or 3 volts, the drop in potential for each of the vats in parallel being barely  $\frac{1}{2}$  volt.

In this process the current density employed was about 2 amperes per square foot, and the deposition of lead was about  $\frac{3}{4}$  lb. per square foot of cathode surface per twenty-four hours. The lead adhered to the cathode plates in loose crystalline masses, which had to be removed by scrapers, and fell to the bottoms of the vats, thence to be shovelled out at intervals after removing the anode and cathode frames.

The *Tommasi* process employs as cathode a series of metallic discs revolving between the anodes and half-immersed in the electrolyte, which is a double acetate of lead and potassium. The spongy lead deposit is scraped off the cathode discs mechanically, when it has become thick enough, washed, compressed, and melted under charcoal.

Neither of the above processes has proved satisfactory on a commercial scale. When only very low current densities were employed the deposit, although loose and spongy, would be fairly regular in thickness, but as soon as the density was raised to such a figure as would give a reasonable output per tank the deposit would become irregular, and short-circuiting would occur. The addition of glue or gelatine to the solution slightly improved matters, but the deposits were always crumbling and loosely coherent; moreover, the solutions employed were so dilute that a great tank capacity was necessary for a comparatively small output.

Commercial electrolytic refining has been made possible through the great discovery of *Betts*—namely, the use of lead fluosulphate as an electrolyte. This salt, besides being soluble in water to any extent, 1 part of water sufficing to dissolve 3.57 parts of the crystallised salt, already containing 4 molecules of water of crystallisation, has a very high conductivity, and is free from secondary decomposition when electrolysed, there being no

\* *Trans. A.I.M.E.*, vol. xii.

polarisation from formation of lead peroxide on the anode, as happens when acetate and nitrate solutions are employed.

**The Betts Process.\***—This is employed at the Canadian Smelting Works, Trail, B.C., at the De Lamar Refinery, South Chicago, and at Locke Blackett's Refinery, Newcastle-on-Tyne. In the early experiments with this process, even with the use of the fluosilicate electrolyte, the deposit was crystalline in texture, and the cathode had to be removed from the tanks at intervals and put through rollers in order to pack down the lead and make it moderately coherent, but the addition of glue or gelatine enabled a smooth and coherent deposit to be obtained, even when the current density was much increased. The electrolyte ordinarily employed contains 5 to 8 per cent. Pb and 10 to 11 per cent. free  $\text{H}_2\text{SiF}_6$ . Without gelatine a current density of 9.6 amperes per square foot gives a very irregular deposit, but with the addition of 0.1 gramme gelatine per litre the deposit becomes smooth, lustrous, and so coherent as to permit of bending without breaking.

The fluosilicate electrolyte is prepared as follows:—Commercial hydrofluoric acid (35 per cent. HF) is diluted with water and passed through a series of shallow tanks. In the first is a layer of pulverised quartz 2 feet thick, through which the solution percolates slowly, forming fluosilicic acid. In the second tank white lead is added and dissolves with effervescence; all sulphuric acid and any hydrofluoric acid which may not have taken up silica here settle out as lead sulphate and fluoride respectively. In the next tank the solution is filtered, and thence runs by gravity into the electrolytic tanks. The strength of solution used formerly was 6 per cent. Pb and 15 per cent. of total  $\text{SiF}_6$ , but is now more commonly 5 per cent. Pb and 8 to 11 per cent. of  $\text{SiF}_6$ . For every ton of refined lead produced  $\frac{1}{2}$  lb. of glue is added, and the temperature is kept at 30° C.

The tanks used at the Trail plant are 7 feet long, 2 feet 6 inches wide, and 3 feet 6 inches deep, made of selected Douglas fir 4 inches thick, except the partition between each pair, which is 5 inches thick. The planks, tongued and grooved, are coated with a  $\frac{1}{4}$  inch layer of asphalt (melting point 40° to 50° C.), and are bolted and tied with iron rods; the sloping floor is covered with P. and B. roofing paper, coated with P. and B. paint, and then covered with  $\frac{1}{4}$  inch of asphalt. All leakage is collected by means of launders in a settling tank. The daily output from each tank is about 570 lbs. of refined lead, making a total for the entire plant of 70 short tons per day.

The anodes are cast by means of a Rösing pump from two 50-ton melting and drossing kettles in moulds like those used for copper refining; they are 26 inches wide by  $3\frac{1}{2}$  inches deep, and about  $1\frac{1}{2}$  inches thick, have two lugs cast on for suspension, and weigh about 300 lbs. each. The cathodes are thin sheets of lead (measuring 36 inches  $\times$  26 inches  $\times$   $\frac{3}{16}$  inch thick), and were formerly prepared by deposition in special tanks on sheet steel cathodes prepared by thoroughly cleaning, flushing with copper, lightly lead plating, and then varnishing with a solution of paraffin in benzine, which, after drying on, allowed the sheets of deposited lead to be readily stripped off. The cathode sheets were deposited in only thirty hours with a current density of 4 amperes. Now, however, they are prepared by casting by means of a

\* A. G. Betts, *Trans. A.I.M.E.*, vol. xxxiv (1904), p. 175; also P. L. Whitehead, *Mines and Minerals*, 1905, xxv., 285. See also Betts, *Lead Refining by Electrolysis*. New York, 1908.

horizontal trough extending along the whole width of a plate inclined at  $15^\circ$  from the horizontal, filling the trough with molten lead, and tilting its contents upon the plate.

Each tank contains 20 anodes and 21 cathodes placed  $1\frac{1}{2}$  inches apart, the distance between anode and cathode being thus between  $1\frac{3}{4}$  and 2 inches. The area of each immersed anode is 11 square feet, the current density employed is 14 amperes per square foot of cathode area, and the fall of potential between anode and cathode is 0.25 to 0.3 volt, this low figure being due partly to the high conductivity of the electrolyte, and partly to the ingenious system of contacts employed, which consists of little wells of mercury in the bus-bars, into which dip on the one side copper pins soldered to the cathodes, and on the other similar pins clamped to the anodes. The cathode sheets are suspended from copper crossbars measuring 1 by  $\frac{1}{2}$  inch. The cathodes are exchanged every four days, the anodes are corroded away in about eight days, and make about 22 per cent. of scrap.

As 1 ampere of current deposits 3.88 grammes of lead per hour, or 3 $\frac{1}{2}$  times as much copper as could be deposited with the same current, a ton of lead requires about 260,000 ampere-hours, which, at 15 amperes per square foot, means about 722 square feet of cathode area per ton of daily output. In practice, owing to short-circuiting, to increased resistance caused by the accumulation of slime upon the anodes, &c., and to other causes, the consumption of current is about 80 per cent. more than the theoretical, but in any case the cost of fuel for producing current is less than the cost of fuel for refining in the dry way.

The precipitated lead is remarkably pure, especially when smoothest, most of the impurities seem to be introduced through the attachment of particles of slime to the surface of the cathodes, resulting in rough lumps, which often contain  $2\frac{1}{2}$  ozs. of silver, as against one-tenth of that amount for the average smooth deposit.

TABLE LIV.—ELECTROLYTIC LEAD REFINING.

	Work-Lead.	Refined Lead.	Anode slimes.	
			1	"
Pb, . . . . . per cent.,	98.1014	..	13.83	11.09
Ag, . . . . . "	0.8934	0.0009	not	31.92
Au, . . . . . "	0.0122	..	given	0.45
Cu, . . . . . "	0.1800	0.0006	15.60	4.00
Fe, . . . . . "	0.0140	0.0012	1.20	undet.
Sn, . . . . . "	0.0289	0.0059	8.91	..
Sb, . . . . . "	0.0076	0.0043	24.13	30.15
As, . . . . . "	0.1693	0.0001	25.60	3.56
Ag, . . . . . ozs. per ton.	260.6	0.27	not	9310
Au, . . . . . "	3.57	..	given	131.16

The cathodes, when removed from the tanks, weigh from 150 to 200 lbs.; they are washed and scrubbed to remove adhering slime, melted down in

a 50-ton kettle, poled, and cast into bars by means of a Rösing lead pump. The general average of a large number of analyses of refined lead by this process, as well as of a similar average of many analyses of original work-lead and of the anode-slimes is given in Table LIV. The poling operation removes most of the tin and residual antimony and arsenic left in the cathode metal.

Of the impurities present in the original work-lead, zinc, nickel, and cobalt dissolve in the solution, as also does a portion of the iron; arsenic, antimony, bismuth, and copper, the electromotive force of all of which is much below that of lead, remain in the slime together with the gold and silver. Tin alone appears to have nearly the same electromotive force as lead, and so partly follows the lead deposited on the cathode, from which, however, most of it can be readily separated by poling. The completeness of the separation of antimony and bismuth is remarkable.

**Treatment of Slimes at Trail.\***—The slimes produced in a plant turning out 70 tons of refined lead per day weigh over 5 tons daily, and their treatment for best results is still somewhat of a problem. Their treatment until recently was as follows:—After scrubbing the anodes to detach the slime, it is stirred up with electrolyte, settled, and the solution returned to the electrolysing tanks. The slime is then washed with three successive quantities of hot water in three tanks one below the other, after which, however, it still retains about 1 per cent. of  $\text{H}_2\text{SiF}_6$ . The wash-waters are evaporated at  $30^\circ \text{B}$  (= 16 per cent. Pb and 11 per cent.  $\text{H}_2\text{SiF}_6$ ) in a copper tank 8 feet  $\times$  8 feet  $\times$  18 inches deep, drawn off into a settling tank, cooled, and gradually returned into the circulating electrolyte.

The slimes now nearly free from acid, were, until recently, boiled with caustic soda in the proportion of 200 lbs. to each 1,000 lbs. of dry slime in steel tanks 8 feet  $\times$  3 feet  $\times$  6 feet deep, with oval bottoms, along which run two perforated steam pipes. After boiling up, the supernatant solution was run off, the slime washed twice with hot water, and sluiced into a wooden tank lined with 12-lb. lead for acid treatment. The soda solution and its wash-waters contained all the antimony, which was present as  $\text{Sb}_2\text{O}_3$  (about half of the total antimony in the slime); it was recovered by electrolysing with a current density of 12 amperes per square foot, using iron or lead cathodes; in the latter case a hard lead was obtained with 25 per cent. Sb.

The slime was next boiled in the lead-lined tanks for eight hours with 15 per cent.  $\text{H}_2\text{SO}_4$  to remove Cu, Fe, and Bi, air being injected through perforated pipes to facilitate solution. The solution and subsequent wash-waters were concentrated, and after crystallising out blue vitriol, the molten liquor was used over again with more free acid.

The slime was then dried in a direct-fired steel pan, 4 feet  $\times$  8 feet  $\times$  1 foot deep, mixed with 400 lbs. of soda ash and 50 lbs. of nitre per ton of slime, and smelted down slowly in a water-jacketed cupelling furnace lined with magnesia brick. After skimming off the slag, which is composed chiefly of sodium antimoniate containing 30 ozs. of Ag per ton and a trace of Au, the temperature is raised to oxidise the residual lead, which, at the same time, carries off any antimony remaining and also part of the copper, this lead slag containing 150 to 200 ozs. of Ag and 0.5 oz. of Au per ton.

After two hours' heating the bullion left upon the test is free from Pb, but is only 900 fine, on account of the copper present. This, however, is

\* Whitehead, *loc. cit.*, abstract in *Min. Ind.*, vol. xiv., p. 425.

removed by further oxidation as  $\text{Cu}_2\text{O}$ , which runs off in a liquid condition like so much litharge, and contains Cu 40 per cent., Ag and Au 3,000 ozs. per ton; it takes three hours to bring up the 40,000 ozs. charge of doré silver from 900 to 950 fine (in other words, to oxidise some 2,000 ozs. of copper present).

At the present time all the above preliminary treatment is dispensed with, and the thoroughly washed slime is loaded into copper cars, run under a drying hood 6 feet wide by 13 feet 10 inches long, which has a hood and sliding doors, and admits two cars at a time. The dried slime is mixed with soda ash and smelted down direct to doré bullion about 950 fine.

The doré silver is cast into 500-oz. bars, and parted with sulphuric acid in the usual way. The silver sulphate is decomposed with metallic copper, and the precipitated silver is melted down upon the magnesia-brick water-jacketed test of another cupelling furnace holding 80,000 ozs., where, after melting, it is covered with charcoal and cast in 1,200-oz. bars. The gold is melted down in a graphite crucible, refined with nitre, and so raised to 995 to 998 fine.

A new process for treating the slimes from the electrolytic process described by Betts\* is conveniently summarised as follows:—†

1. Slime is agitated by means of steam and air in a lead-lined tank with ferric sulphate solution, which sulphatises Cu, Pb, and some Ag, the first and last being dissolved, while the second remains in the residue, converts As, Sb, and Bi into oxides, and produces  $\text{FeSO}_4$  and free  $\text{H}_2\text{SO}_4$ . The products are a solution (2) and a residue (3).

2. The solution is treated with metallic copper, precipitating Ag, which is added to residue 3, and a silver-free solution to 5.

3. The residue is filter-pressed and washed, then agitated with a solution containing 12 to 15 per cent. HF and 2 to 4 per cent.  $\text{H}_2\text{SO}_4$ , yielding parting material 4 and solution of  $\text{SbF}_3$ .

4. The parting material is melted with soda and cast into anodes, which are parted by electrolysis in an electrolyte composed of methyl-sulphuric acid (15 per cent.  $\text{CH}_3 \cdot \text{SO}_3$  and 7.5 per cent. Ag, with 1 part of gelatine in 12,000 to 15,000), giving doré silver and an electrolyte containing Cu, Bi, and about 1.5 per cent. Ag, from which the Ag can be removed by Cu, the Cu and Bi with Pb, the Cu and Bi being then separated by boiling with  $\text{Fe}_2(\text{SO}_4)_3$ , the residual  $\text{Bi}_2\text{O}_3$  after washing being reduced to metal.

5. To the solution  $\text{CuO}$  is added,  $\text{As}_2\text{O}_3$ , if present, is crystallised out, and it is then electrolysed for Cu with carbon anodes, the ferric sulphate solution remaining being returned to (1).

6. The solution of  $\text{SbF}_3$  from (3) is electrolysed, yielding metallic Sb and dilute HF, which, with  $\text{H}_2\text{SO}_4$ , is returned to (3).

The following analyses are given of the products from this new process:—Metallic antimony Sb 99.52 per cent., Cu 0.07 per cent., As 0.41 per cent., doré silver Ag 78.94 per cent., Au 2.08 per cent., Pb 17.56 per cent., Cu 0.81 per cent., Sb 0.47 per cent., As none.

Space is wanting here to go into the subject of electrolytic refining in greater detail, and for further information the work on the subject by Betts‡ should be consulted.

\* *Electrochemical and Metallurgical Industry*, 1905, vol. iii., p. 272.

† *Mineral Industry*, vol. xiv., p. 427.

‡ *Lead Refining by Electrolysis*. New York, 1908.

## SECTION IV.—SUPPLEMENTARY.

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### CHAPTER XIX.

#### WORKS ASSAYING AND ANALYTICAL METHODS.

**Preparation of Samples.**—The last stage in the preparation of an ore sample, after a series of quarterings which have reduced its bulk to about a couple of pounds, is the fine grinding. Previous to commencing this, it is well to dry the final sample, after which it is ground down on a clean bucking plate till it will pass the final sieve adopted, which for ordinary lead and medium grade silver ores may be an 80 or 100 mesh, but for rich gold or silver ores should be 120 mesh, any coarse particles being returned and re-ground until they are got to pass the sieve. The whole sample is then thoroughly mixed by rolling over and over alternately in two directions at right angles to each other on a sheet of oilcloth or thin india-rubber, after which it is filled into a series of from four to eight bottles or stout paper bags placed in a row, by means of a funnel partly stopped by the forefinger, and in such a way that part of the sample enters each bottle of the row successively, the funnel travelling up and down the row till all are filled simultaneously. One sample is then delivered to the works' assayer and two to the seller's representative, the others being retained as "checks" or "controls" in case disputes should arise.

In case part of the sample remains upon the sieve as metallic threads or scales, which frequently happens in the case of ores containing native silver or "chlorides," and occasionally with rich gold ores or concentrates such portion is weighed and assayed separately, the weight of sample passing the sieve being also determined. The correct assay value of the ore is then the amount shown by assay of the fine powder *plus* the assay of the "metallics" calculated back on the weight of sample which passed the sieve.

#### ASSAYING.

Strictly speaking, the term assaying in smelting works is confined to the determination of the lead, silver, and gold contents of an ore as a basis for purchase, and as regards rich galenas little else is ever required. In the case of complex ores, however, it is necessary to determine many other ingredients, not only as a guide in making up the furnace charges, but also as a basis for purchase, as has been explained (Chap. ix.).

The determination of lead, silver, and gold as a basis for purchase is always

made in the dry way—all other constituents are determined in the wet way by short and convenient methods, most of them volumetric. These are described under the head of *Analytical methods*.

**Dry Assay for Lead.\***—There are many different modifications of the dry assay for lead, but only the two which give most satisfactory results need be described here.

1. *Method for Pure Galenas containing upwards of 70 per cent. Lead.*—The best method is the *iron crucible assay*, commonly used in England, and in Belgium and Silesia; it is performed in the ordinary crucible wind-furnace, and the crucible employed is of wrought iron 4 inches high. The quantity of ore taken is usually 400 grains or 30 grammes (at Tarnowitz 50 grammes), and the flux employed is, according to Percy, a mixture of carbonate of soda 7 parts, dried borax 3 parts, and cream of tartar 1 part,† an equal quantity of which is mixed with the weighed sample. The iron crucible is brought to a bright red heat, the mixture shot in from a copper scoop, and the crucible replaced in a hot fire without loss of time, a “cover” of borax being added. As soon as the mixture is completely melted (which should take about five minutes), the contents of the crucible are stirred and its sides scraped down with a chisel-ended iron rod. The crucible is returned to the furnace for a few minutes, again removed, the sides again scraped down and the stirrer washed in the slag, after which the crucible is tapped a few times to settle the lead, and poured; any small globules adhering to the pot being also turned out by a sharp tap or two of the crucible held upside down in the tongs. When cool, the button of lead is separated from the slag (which is generally quite clean), cleaned, and “cubed up” under the hammer, and weighed. Pure galenas, by this method, will always yield 84 per cent. of lead, provided the temperature is high enough to complete the assay in ten minutes, whereas with the “iron-nail” and other methods it is impossible to get more than 82 or 82½ per cent. Percy proved‡ that when the gangue consists exclusively of calcite or barytes the results given by this method are not materially lower in the case of even very poor ores than with rich, the actual loss averaging only 1 to 1½ per cent. on the weight of ore taken. When, however, the gangue consists mainly of silica the results on poor ores (8 to 25 per cent.) are from 3 to 3½ per cent. lower than the truth, this error in the case of an 8 per cent. ore amounting to no less than 35 per cent. of the total lead present.

2. *Method for every other Variety of Lead Ore.*§—Many different methods of assay in clay crucibles have been recommended in works on assaying, and the number of fluxing mixtures employed is very large, but according to pretty general experience no method is so universally applicable as the

\* The student may consult Phillips' *Elements of Metallurgy*, 1891, pp. 626, *et seq.*; Percy, *Metallurgy of Lead*, pp. 104-119; Balling, *Probirkunde*, Brunswick, 1879, and Gautier's translation, *Manuel de l'Art de l'Essayeur*, Paris, 1881; Mitchell's *Manual of Assaying*; Beringer, *Text-book of Assaying*, 1906, pp. 211-213; Fulton, *Manual of Fire Assaying*, N.Y., 1907, &c.

† The author has found that ordinary “lead flux,” composed as described in the next section, can be advantageously substituted for this mixture.

‡ *Metallurgy of Lead*, pp. 113, *et seq.*

§ v. Furman, *Manual of Practical Assaying*, N. Y., 1893, p. 137; also Hofman, *Metallurgy of Lead*, N. Y., 1893, p. 70; Iles, *S. M. Q.*, vol. xv., p. 336; Fulton, *Manual of Fire Assaying*, N. Y., 1907.

so-called "Colorado assay," which is practically identical with the Freiberg assay described by Balling,\* and was undoubtedly brought from Freiberg. The assay is conducted in the ordinary large scorification muffle, which is 19 inches by 12 inches inside, and the crucibles used are the "Colorado A A and A" of the Battersea Crucible Company, the former (corresponding with the 5-gramme cup of the Denver Fireclay Co.) being  $2\frac{1}{2}$  inches high by  $2\frac{1}{4}$  inches wide at top, and the latter 3 inches high by  $2\frac{3}{4}$  inches wide (corresponding with the 10-gramme cup of the Denver firm). The smaller size is amply sufficient for all lead ores properly so called—i.e., over 10 per cent. lead contents—while the larger size is required for slags and poorer ores. There is no agreement about the best composition for lead flux, but in the author's experience good results were obtained by adding to 5 grammes of ore, 20 grammes of flux made according to Plattner's original formula (with the exception that dried carbonate of soda is substituted for the bicarbonate originally specified)—viz., 2 parts potassium carbonate, 2 parts sodium bicarbonate, 1 part powdered borax glass, and 1 part flour. Furman,† however, recommends a lead flux with only half the above proportion of borax, while Iles ‡ uses 30 grammes of a flux made up by the Denver Fireclay Company, in which the proportion of sodium carbonate is somewhat increased. The ore and flux should be very thoroughly mixed in the crucible with a spatula or glass rod, and well settled down by tapping. According to Iles, a cover of borax is only requisite in the case of slags and basic ores, but many assayers use it in all cases, except in presence of very large amounts of As, Sb, and Zn, and its use generally gives higher results, though it undoubtedly prevents the elimination of volatile metals. After well tapping down, from 1 to 4 "tenpenny" wire nails are added, points downward, for in that way they are upon removal less likely to retain globules of lead than if heads downward, as recommended by Mitchell and the older assayers.

No nails are required if the ore be oxidised; one is sufficient in the case of a slag, three if a rich sulphide ore, and four if pyritous or cupriferous. Fusion in the muffle takes from fifteen to twenty minutes if the heat be right, or may take thirty minutes if the temperature be too low. As soon as tranquil fusion is obtained the crucible is withdrawn, and the nails removed one by one with a pair of short tongs, washing each carefully in the slag to detach any adhering globules of lead. The crucible is then tapped a few times, and poured into a scorifier mould, or on to a cast-iron plate as recommended by Iles. In the latter case the button cools almost instantaneously, and is readily separated from the brittle slag, which should contain no globules. Duplicate assays ought to agree to within 0.5 per cent., and in practice they frequently agree within 1 centigramme on the button (= 0.2 per cent.).

In spite of this agreement, however, and of the fact that lead ores are always purchased on the dry assay, the method is not accurate; for there is always a large though *variable* volatilisation loss, which is compensated to a *variable* extent by the presence of other metals alloyed with the button. Iles § has gone into this matter in detail, determining the impurities in 700 buttons from ores averaging 34 per cent. lead, and checking the dry assay by the wet assay of Alexander given below. He finds the average volatilisation

\* French edition, *Manuel de l'art de l'Essayeur*, 1881, p. 444.

† *Op. cit.*, p. 68.

‡ *Sch. of Mines Quarterly*, vol. xv., p. 341.

§ *Sch. of Mines Quarterly*, vol. xv., p. 339.



loss on ores of this grade to be 7.20 per cent., which is offset by impurities in the buttons amounting to 3.75 per cent. of their weight (gold and silver 0.42 per cent., Sb, Cu, Fe, &c., 3.33 per cent.); so that the net amount by which the dry assay is too low, is in this case 3.45 per cent. of the weight of lead present. He suggests the employment of the wet assay for lead, with a uniform deduction of 2 per cent.; as is nowadays commonly done in the case of copper, to obtain the so-called "dry assay" on mattes and high grade ores. Warwick\* also finds that the wet methods give results which average about 2 per cent. higher than dry assays on ores carrying from 10 to 14 per cent. lead.

Lay† has investigated the muffle dry assay for lead as ordinarily carried out, and found that by starting at a low temperature and raising the heat as quickly as possible practically all the lead which does not enter the button can be found in the slag, the volatilisation loss being, therefore, quite low, which one would perhaps not have expected.

The use of KCN as an addition to the ordinary charge has been advocated by Frost,‡ who adds 5 grammes of cyanide to the charge after complete fusion has taken place, and keeps the crucible in the fire for another fifteen minutes before pouring. The residual lead, which by the ordinary method remains in the slag, is reduced to metal by this addition, and the results average about 0.7 per cent. higher than by the ordinary method, so that on rich galena ores they check fairly well with the wet assay, reduced impurities entering the lead in about sufficient quantity to compensate for the volatilisation loss.

**Wet Assay for Lead.**§—There are several excellent methods of determining lead in the wet way, but for works' purposes only those which are quick as well as accurate are acceptable. The following well-proven methods are in common use:—

1. *Gravimetric, weighing as Pb (von Schultz & Low).*||—Take 1 gramme of ore, dissolve in a 250 c.c. flask with 10 c.c. concentrated  $\text{HNO}_3$  and a few drops of  $\text{HCl}$ , boil, add 10 c.c. strong  $\text{H}_2\text{SO}_4$  and boil till white fumes of  $\text{SO}_3$  are given off. Cool, dilute with 50 c.c. of water, add 2 grammes Rochelle salt, boil, filter, and wash residue of  $\text{PbSO}_4$  and gangue with 1:10 dilute  $\text{H}_2\text{SO}_4$ . Dissolve out the  $\text{PbSO}_4$  on the filter into a small beaker with a boiling solution of  $\text{AmCl}$ , decant into original flask, boil residue with a few c.c. of strong  $\text{NaHO}$ , precipitate with  $\text{H}_2\text{SO}_4$ , avoiding excess and rinse into beaker; acidify with  $\text{HCl}$  and add two or three strips of  $\text{Al}$  sheet¶ to precipitate the lead. When precipitation is complete (five minutes is generally sufficient, but it is safer to allow fifteen) boil a few moments, fill up flask with cold water, settle and decant into a porcelain dish to make sure of catching any particles of lead, repeat the washing several times, then invert over a tared

\* *Proc. Colo. Scient. Soc.*, vol. vii., pp. 73-77.

† *Journ. Can. Mining Inst.*, 1901, p. 224; abstract in *Min. Ind.*, 1901, vol. x., p. 419.

‡ *E. and M. J.*, May 17, 1902, p. 731.

§ *E. and M. J.*, June 18, 1892, and *J. Am. Chem. Soc.*, vol. xv., p. 548; Beringer, *Text-book of Assaying*, 1906, pp. 213-218; Rhead and Sexton, *Assaying and Metallurgical Analyses*, 1902, pp. 62-67.

|| *v. Furman, op. cit.*, p. 137, *et seq.*; also Hofman, *Min. Ind.*, vol. iii., p. 422; and Low, *J. Am. Chem. Soc.*, vol. xv., p. 548.

¶ Zinc free from lead will do, but is much slower than the aluminium sheet, which should be  $\frac{1}{8}$  inch thick, and cut into fragments of about 1 inch  $\times$   $\frac{3}{8}$  inch.

porcelain crucible to collect the lead, rub off the Al with the fingers, settle and decant off the water. Press the Pb into one mass with an agate pestle, wash with hot water and then with 95 per cent. alcohol, dry carefully and weigh quickly. The determination takes from thirty to forty minutes, and the results are accurate, though subject to a *plus* error of from 0.1 to 0.5 per cent., due to impurities (chiefly silicon) in the aluminium, which are weighed as Pb; and to some oxidation of the lead itself. The method has been in common use in laboratories of works under the author's control, and although somewhat slow, has been found very reliable.

2. *Volumetric, with Standard Ammonium Molybdate (Alexander).*\*—Ammonium molybdate gives a white precipitate with lead acetate until the whole of the lead present is thrown down, any excess of molybdate being shown at once by the yellow colour which it gives with a fresh solution of tannin. The indicator is made by dissolving 1 gramme of tannin in 300 c.c. of water, and is used in the form of drops on a porcelain plate. The standard solution is prepared by dissolving 9 grammes  $\text{Am}_2\text{MoO}_4$  in 1 litre of water containing a few drops of  $\text{AmHO}$ . One c.c. of the solution should equal about 0.01 gramme of lead, but it is standardised by weighing out 0.3 gramme of pure  $\text{PbSO}_4$ , which is dissolved in hot ammonium acetate, acidified with acetic acid and diluted to 250 c.c. It is then heated to boiling, and the molybdate dropped in till a yellow colour is produced on adding a drop of the solution to one of the tannin drops on the plate. The standardisation is performed in duplicate, and the value of 1 c.c. calculated.

For the assay, take 1 gramme if under 30 per cent. Pb, or  $\frac{1}{2}$  gramme if over that percentage, heat in a small porcelain dish with 15 c.c. strong  $\text{HNO}_3$  and 10 c.c. strong  $\text{H}_2\text{SO}_4$  until the former is all expelled and white fumes of  $\text{SO}_3$  begin to come off. Cool, dilute with a few c.c. of cold water, add hot water and boil out soluble sulphates; decant into a small filter, wash in the dish thrice with hot dilute  $\text{H}_2\text{SO}_4$  (1 : 10), and once with a little cold water. Boil up with saturated ammonium acetate, pour through filter into a small beaker; repeat till  $\text{PbSO}_4$  is all dissolved, wash with hot water, acidify with acetic acid, dilute with hot water to 250 c.c., and heat to boiling. Run in the molybdate with constant stirring, testing drops on the plate with the tannin indicator. The accuracy of the method is not interfered with by As, Sb, Zn, Cu, or P, which all pass into the original  $\text{H}_2\text{SO}_4$  solution; Low, however, states that it is interfered with by lime.

3. *Volumetric with Standard  $\text{K}_4\text{FeCy}_6$ .*—This method, as described by Beebe,† is carried out in an acetate solution free from alkali salts, uranium acetate, acidified with acetic acid, being used as indicator. The standard ferrocyanide solution is 11 grammes per litre. Either 0.5 gramme or 1.0 gramme is taken and dissolved in  $\text{HNO}_3$  with a little  $\text{HCl}$ , if necessary. Evaporate with  $\text{H}_2\text{SO}_4$  to strong fumes of  $\text{SO}_3$ , dilute a little, and cool. Add an equal volume of alcohol, shake, allow to stand, filter and wash with hot water acidified with  $\text{H}_2\text{SO}_4$ . Wash residue from filter with 50 c.c. of cold concentrated solution of ammonium carbonate, stir vigorously at intervals during fifteen minutes, filter and wash residue thoroughly with hot water. Dissolve out the lead carbonate in hot dilute acetic acid, cool, and titrate.

\* *Colorado Sch. of M. Scient. Quarterly*, vol. ii., No. 1, p. 60; also *E. and M. J.*, April 1, 1893.

† *E. and M. J.*, July 7, 1894.

Standardise with pure lead acetate. The method is fairly accurate, but the manipulations are somewhat tedious for everyday use.

4. *Volumetric Methods with Standard  $KMnO_4$ .*—In view of the great convenience of titration with permanganate which requires no indicator, a number of methods have been introduced for utilising this solution.

The original method of Knight, as modified by Low,\* consisted in decomposing, extracting the  $PbSO_4$  and precipitating the Pb or Al exactly as in Low's gravimetric method. He then dissolves the Pb in 5 c.c.  $HNO_3$  (1 : 2), warms, pours into a small beaker to get out the Al, returns to flask, adds 2 drops of phenolphthalein solution, then a slight excess of strong NaHO, and lastly, 10 c.c. cold saturated solution of oxalic acid, cools, filters, and washes with cold water. Place in flask 75 c.c. of hot water, add a few drops  $H_2SO_4$ , boil, drop in filter with contents, and titrate with decinormal permanganate.

This is an admirable method, subject only to a *minus* error amounting to about  $\frac{1}{2}$  per cent., owing to the solubility of lead oxalate. Its great convenience is that it does away with the necessity of keeping a separate and expensive standard solution of molybdate, and utilises the same solution already employed for volumetric determinations of manganese, iron, and lime, its standard in terms of Pb being that in terms of Fe  $\times 1.848$ . In a works' laboratory it is often a convenience to employ as few standard solutions as possible, since the amount of standardisation to be done is thus reduced to a minimum. The method, however, is not nearly so direct and rapid as that with molybdate, which is, therefore, to be preferred in works where large numbers of wet lead assays have to be made.

Another method proposed by Low† for utilising the permanganate titration involves precipitation of the lead as chromate after boiling out the sulphate with lead acetate. The lead chromate is boiled with hot oxalic acid and alcohol to reduce  $CrO_3$ , by which the lead is converted into oxalate, which is then filtered off, well washed with hot water, boiled up paper and all with dilute  $H_2SO_4$ , and titrated with standard permanganate solution of 1.5185 grammes per litre, of which 1 c.c. = 1 per cent. Pb when 0.5 gramme of the original ore was taken.

Still another method recently published‡ consists in precipitating the lead as  $PbO_2$  from its solution in strong NaHO by means of bromine water. The filtered and washed  $PbO_2$  is then reduced by a measured excess of standard ferrous-ammonium-sulphate solution, and the excess of this titrated back by permanganate.

*Determination of Lead in Slags.*§—Take 10 grammes slag, add 10 c.c. water, and, while agitating, pour in 50 c.c. concentrated HCl, dilute with 30 to 40 c.c. water, and boil till no more  $H_2S$  comes off. Add  $KClO_3$ , boil off excess of acid, filter into a beaker holding 1,500 c.c., and wash three times with boiling water. Return filter with residue, boil up again, filter again, and wash twice or thrice with boiling water. Add 20 grammes granulated zinc free from lead, allow to stand overnight, filter, wash three times with hot water. Take filter with excess zinc and precipitated Pb in a beaker,

\* *Journ. Am. Chem. Soc.*, vol. xv., p. 548; abstract in *Min. Ind.*, vol. iii., p. 422.

† *Journ. Am. Chem. Soc.*, vol. xxx., p. 587; and *Min. Scient. Press*, Feb. 13, 1909, p. 258.

‡ *E. and M. J.*, Jan. 30, 1909, p. 260.

§ Svoboda, *Oest. Zeitschr. f. B. u. H. Wesen*, 1902, p. 583.

add  $\text{HNO}_3$  to dissolve, transfer to 250 c.c. flask, cool, filter to mark. Take 200 c.c. of filtrate (= 8 grammes slag), add 25 c.c. of 1 : 4  $\text{H}_2\text{SO}_4$ , and, when  $\text{PbSO}_4$  has settled out, add 70 c.c. absolute alcohol, and allow to stand overnight. Filter through Gooch crucible, wash with 30 per cent., and then with 100 per cent. alcohol, finally with ether, dry at  $150^\circ \text{C}$ ., cool and weigh. The  $\text{PbSO}_4$  is usually chemically pure, but may be tested by dissolving out with ammonium acetate and weighing the residue, if any.

**Assay for Silver and Gold.\***—These metals are always determined by the dry assay, according to which they are first collected in a button of lead, which is then cupelled and the resulting bead of gold and silver weighed. The gold is afterwards “parted” from the silver by means of nitric acid, and weighed in the metallic condition; the difference between its weight and that of the bead obtained by cupellation being the weight of silver.

In the case of rich galenas, of which 400 grains or more have been taken for lead assay by the iron crucible method, the reduced button of lead contains all the values in gold and silver, and all that is necessary is to cupel this button.

For all other ores a separate assay must be made for the precious metals. Two different methods are employed—viz., the *crucible assay* and the *scorification assay*. The former is in use everywhere for gold ores, but for silver chiefly in England and on the Pacific Coast; while the latter is exclusively employed for lead-silver ores in Germany as well as in Colorado and other smelting centres of the United States. When the ore to be assayed is either poor or refractory (especially if basic); when a large part of its silver contents is in the form of chloride or combined with arsenic or antimony; or when it contains any mineral which even in fine powder decrepitates on heating, the crucible assay will generally give more uniform and more reliable results. The scorification assay is undoubtedly to be preferred for lead ores, for coppery ores, for sulphides (except such as decrepitate badly), and generally for those medium grade docile ores which contain most of their silver finely disseminated through the other minerals. On such ores the uncorrected scorification assay is higher than the uncorrected crucible assay.

According to Nissenon† assays made of Broken Hill concentrates at Stolberg are higher than those made by cupelling the lead buttons obtained by the wrought-iron crucible dry assay for lead by no less than 3·8 per cent., but an examination of the silver buttons showed that those from scorification assays contained on an average 2·15 per cent. impurities, as against only 0·52 per cent. in the buttons from the crucible assays, thus accounting for 1·53 per cent. out of the total surplus of 3·8 per cent. Prost and Boveroulle‡ confirm these results, giving 2·63 per cent. as the amount by which the scorification assay exceeds the iron pot crucible assay.

**Scorification Assays.**—These are best performed in triplicate, in a muffle sufficiently wide to take three of the largest-sized scorifiers side by side. The muffles used in American smelting works are usually 19 inches by 12 inches wide, and they are generally heated with bituminous coal (Swansea

\* See Beringer, *Text-book of Assaying*, 1906, pp. 88-116; Rhead and Sexton, *Assaying and Metallurgical Analyses*, 1902, pp. 115-121; Fulton, *Manual of Fire Assaying*, N. Y., 1907, &c.

† *Berg. u. H. M. Zeitung*, vol. lix., 1900, p. 572.

‡ *Revue Universelle des Mines*, 1901, vol. lvi., p. 220.

fashion) instead of with coke. The scorifier commonly used in Colorado is the  $2\frac{1}{4}$ -inch size, the charge for which is  $\frac{1}{10}$  assay ton (or on the 2,000 lbs. ton scale, 2·9167 grammes, upon which quantity each  $\frac{1}{10}$  mg. of silver obtained represents 1 oz. per ton). With ores which are very refractory, or which contain much of volatile constituents, a larger sized scorifier is necessary to leave room for the extra test lead and borax required; and some assayers habitually use the  $2\frac{1}{2}$ -inch scorifier for poor ores and take  $\frac{2}{10}$  of an A.T. for each assay so as to get larger beads to "part" for gold. When only  $\frac{1}{10}$  A.T. is taken for silver it is usual to make a separate crucible assay for gold, taking  $\frac{1}{2}$  A.T. each if performed in the muffle, or 1 to 2 A.T. if in the wind furnace.

The following table (substantially after Kerl) gives the proportions of granulated lead and of borax glass for  $\frac{1}{10}$  A.T. of material, but a little roll of sheet lead or paper packet of borax glass can be added after scorification has commenced, when necessary. It is in fact advisable not to add at once all the borax glass, as that would result in "covering" the lead too soon; it is preferable to start with a small quantity, say  $\frac{1}{4}$  gramme, and add more afterwards as required.

TABLE LV.—CHARGES FOR SCORIFICATION ASSAY.

Nature of Material.	Grms. of Test Lead.	Grms. of Borax Glass.	Temperature.	Advantageous Additions, &c.
Galena, . . . . .	10-15	0·0-0·5		5 grms. litharge helps.
Galena, with blende and pyrites, . . . . .	20-35	0·6-0·9		5 grms. litharge helps.
Lead carbonate, . . . . .	10-15	0·0-0·5		
Iron pyrites, . . . . .	30-45	0·3-0·8		
Arsenical pyrites, . . . . .	45-50	0·3-1·5	High	Cover of litharge.
Antimonial ores, . . . . .	50	1·0-3·0	High	
Blende, . . . . .	30-45	0·3-0·6	High	Iron oxide helps.
Lead matte, . . . . .	25-35	0·5-1·0		
Copper ores and matte, . . . . .	35-50	0·3-1·5	Low	Cover of litharge.*
Speiss, . . . . .	30-60	0·5-0·7	High	
Furnace accretions, . . . . .	25-50	0·3-1·5		
Tellurides, . . . . .	50-60	0·3		Cover of litharge.*
Siliceous ores, . . . . .	35-40	None.		
Basic ores, . . . . .	35-40	0·0-2·0		Powdered glass helps.
Basic ores with barytes, . . . . .	25-30	0·5-1·5		Sodium carbonate helps.
Cupel bottoms, . . . . .	20-25	up to 0·75		

The quantities of borax glass and test lead in the above table are never weighed out; the former being guessed with sufficient accuracy, while the latter is measured in an adjustable measure like that used by sportsmen for loading cartridges. The scorification process is conducted as follows:—

One-half of the test lead being placed in the scorifier, the ore is added and well mixed with a spatula; the other half is then added as a cover, and the borax glass on top of all. The scorifiers are placed in order in the muffle, and the door closed till the formation of a ring of slag round the surface of

\* Button has often to be scorified with more lead.

the melted lead shows the commencement of scorification. The door is then opened and scorification continued until the "eye" of lead is completely closed by a uniform surface of slag; upon which the muffle is again closed and the temperature raised as high as possible for a few minutes, so as to render the slag perfectly fluid. Some teachers of assaying recommend "cleaning the slag" by the addition of a few grains of charcoal or anthracite wrapped in tissue paper before the final increase of temperature, but, providing the slag is perfectly fluid, the omission of this precaution is not found to make any appreciable difference in the result. When thoroughly fluid, the scorifier is removed from the muffle and its contents are poured into the mould, which is preferably conical in shape; when cold the lead button is easily separated from the brittle slag, cleaned by hammering and brushing, and "cubed up" ready for cupellation. The slag should show no half-fused portions, and the lead should always be collected in a single malleable button, which should weigh between 8 and 12 grammes; if heavier it should either be reduced by scorification or cupelled in an extra large cupel. It is difficult to say which practice is preferable, but the loss of silver in scorification is commonly supposed to be less than in cupellation.\*

If the bullion be hard or brittle it must be re-scorified, with the addition of more lead. If the button shows traces of regulus on the top the assay should be rejected and another made, adding a little litharge. If the slag shows signs of imperfect fusion the assay must be repeated, crushing the ore a little finer than before, and mixing it more thoroughly with the test lead, using a little extra borax and a higher heat. The buttons of lead are then cupelled and the beads of silver weighed, the maximum difference between the three should not be more than  $\frac{1}{10}$  mg. on buttons weighing 1 centigramme each (1 oz. on 100 ozs.). When the difference between the heaviest and lightest of the three does not exceed this amount their weight is added together and the mean taken; but when the difference is greater a couple more check assays should be made. All the silver beads obtained are added together for parting unless the gold is to be determined by a separate crucible assay, and in any case the amount of gold found is deducted from the combined weight of the beads to get at the silver present. The amount of silver in the test lead should be always determined by scorifying down and cupelling 100 grammes, and the amount, if considerable, deducted from the weight of the silver beads. When, however, "pure" test lead is used the + error introduced by making no deduction for its silver contents is on all medium-grade ores very much less than the — errors through losses in scorification and cupellation, so that unless the scorifier slag and cupel bottom be re-assayed, and the amount added to that found in the original assay (which is not usually done in ordinary commercial work), a closer approximation

\* Doubt has been thrown on the accuracy of this belief through the researches of Mason and Bowman (*Journ. Amer. Chem. Soc.*, 1894, vol. xvi., p. 313). These authors found that pure silver and gold with test lead almost invariably lost more weight when scorified down and cupelled than when the same weight of test lead was cupelled off direct. With weights of 150 to 600 mgs. the average loss by cupellation alone was 1.99 per cent., by combined scorification and cupellation 2.54 per cent. These results are confirmed by those of Miller and Fulton (*S. M. Q.*, vol. xvii., p. 163). According to Percy (*Met. of Silver and Gold*, p. 296), the late Richard Smith found no difference in the quantity of silver obtained from test lead by direct cupellation and by cupellation after a previous scorification. The subject apparently requires further investigation.

to the truth is obtained by making no deduction for the silver contained in the granulated lead.

*Crucible Assays.*—The special applicability of this method to certain classes of ores has been already noted. Many different fluxing mixtures are in use, but among the best are those given in the following table. Careful examination of the sample with a lens and by washing a little of it in a small porcelain dish or saucer will usually give sufficient information as to its composition to serve for fluxing, which can easily be done to suit; bearing in mind that soda is a flux for silica, borax and powdered glass for iron and other bases; that sulphur in the ore acts as a reducer, but is counteracted by nitre as well as by oxides of iron and manganese.

There are two principal systems of fluxing; the old system advocated by Mitchell aims at keeping an excess of litharge always present to flux base metals, by which means the slags are always fluid at a low temperature, but the destruction of pots is very great, and the proper weight of lead button can in the case of many ores only be obtained after one or more preliminary trials. Not only so, but the presence of a great excess of litharge is invariably found to give low results as regards silver, though probably not as regards gold.\* In the more modern system of fluxing practically the whole of the litharge added is reduced, and the base metals are kept in solution in the slag either as silicates or sulphides. The button by this method is always obtained of the correct weight, but it may contain copper, antimony, or other impurity, in which case it should be scorified, before cupelling, with a little lead, and should any matte or speiss be found overlying the button this should be also in such a case scorified together with the lead.

Table LVI. gives a number of mixtures for conducting crucible assays, which have been tried by experience, and can be thoroughly recommended. All but Nos. 1 and 5 are on the "total reduction system."

Pyrites, mispickel, and other sulphide ores may be oxidised by nitre as with mixture 5; a large pot must be used or the mixture is liable to boil over, but when a muffle is handy (as is usually the case) the author prefers to place the weighed sample in a large roasting dish just inside the muffle door and to roast out the sulphur and arsenic, melting the residue with a mixture intermediate between 1 and 8, according to the proportion of waste in the sample. Slags and poor iron fluxes require at least two assays of 1 ton each to be taken, adding the buttons together for weighing, and, in the case of very poor slags, some assayers even make duplicates of 2 tons each, using slightly larger crucibles, although this is much more troublesome.

Steel † has described a general method of calculating crucible charges which will serve for any class of ore. The material for assay is closely inspected, and its approximate mineralogical composition estimated, the corresponding estimated weight of each ingredient in the quantity of ore taken for assay being written down (say  $\frac{1}{2}$  A.T. equals about 15 grammes). The slag aimed at is a bisilicate of soda and lead, and the calculation is as follows:—For every 1 gramme of  $\text{CaCO}_3$ ,  $\text{MgCO}_3$ , or  $\text{BaSO}_4$  present in the ore, and for every  $1\frac{1}{2}$  grammes of  $\text{FeO}$ ,  $\text{ZnO}$ , clay, or other infusible silicate present, 2 grammes of  $\text{SiO}_2$  should be allowed, and, if not present in the ore, the deficiency should be made up by powdered quartz or powdered glass

\* See also Lodge, *Trans. I.I.M.E.*, vol. XXXIII., 1877, pp. 414-416.

(of which  $1\frac{1}{4}$  grammes is taken as equal to 1 gramme  $\text{SiO}_2$ ). According to the total quantity of  $\text{SiO}_2$  now present in ore and fluxes  $\frac{1}{3}$  gramme borax glass and  $1\frac{1}{4}$  grammes of sodium bicarbonate should be added for each gramme of  $\text{SiO}_2$  present originally or added. Litharge is then added enough to make with the lead in the ore an 18-gramme button, and in addition 2 grammes  $\text{PbO}$  for each gramme silica or infusible silicate present and more, if there is much copper. Argol or nitre should be used according to the amount of sulphur present, so as to get a button of about 18 grammes, and if the resulting button is brittle from antimony or zinc, or hard from much copper, it must be scorified.

TABLE LVI.—CHARGES FOR CRUCIBLE ASSAYS.

	General Formulæ.		Quartzose Ores.		Pyritous Ores.		Basic Gold Ores.	Roasted Concentrates.	Slag.	Poor Siliceous Copper Ore, little Sulphur.
Reference, . .	1	2	3	4	5	6	7	8	9	10
Ore, . . . .	1	1	1	1	1	1	2	1 or 2	2	2
Litharge (red lead), . .	5	$1\frac{1}{2}$	1	1	8	$1\frac{1}{2}$	1	1	$\frac{1}{2}$	$1\frac{1}{2}$
Charcoal, . .	..	..	$\frac{1}{10}$	..	..	..	..	..	..	..
Flour, . . .	..	$\frac{1}{10}$	..	$\frac{1}{12}$	..	..	$\frac{1}{10}$ to $\frac{1}{4}$	..	..	$\frac{1}{4}$
Lead flux, . .	..	..	..	..	..	1	..	1	2	..
Soda carb., .	1	3	$\frac{3}{4}$	2	2	2	1	1	1	3
Borax glass, .	1	$\frac{1}{2}$	$\frac{1}{4}$	$\frac{1}{4}$	1	..	1	$\frac{1}{2}$	1	$\frac{1}{2}$
Powdered glass, .	..	..	..	..	1	..	1	1	..	..
Silica, . . .	..	..	..	..	..	1	..	..	..	..
Nitre, . . .	..	..	..	..	$\frac{1}{4}$	..	..	..	..	..
Iron nails, . .	..	1 to 3	..	1	1 to 3	3 to 4	..	1 or 2	1	1
Cover, . . .	salt	salt	borax	..	salt	salt	borax	borax	borax	borax
Remarks, .	a				b	c	d	e		

*References.*—1. Mitchell, *Manual of Assaying*, 1881, p. 546. 2, 4, and 5. Aaron, *Assaying*, Part I., San Francisco, 1889, pp. 65, 68, 62. 3. Percy, *Metallurgy of Silver and Gold*, p. 245. 6. Furman, *Manual of Practical Assaying*, N.Y., 1893, p. 129. 7, 8, 9, and 10. Author, 1889-1893.

*Remarks.*—a. Flour or nitre to be added according to proportion of sulphur present. b. Without previous roasting. c. After previous roasting. d. The proportion of flour varies according to the nature of the base with lime and alumina one-tenth part is sufficient, with all iron one-fourth part is required, and even more with ores containing manganese oxides. e. When thoroughly roasted to  $\text{Fe}_2\text{O}_3$  about  $\frac{1}{10}$  part of flour is required for reduction.

The crucibles should be heated up before use in the same "nests" in the furnace which they will subsequently occupy during the fusion; if taken out with care they can, after charging, be replaced in the hot coke up to within an inch or two of the top without difficulty. It is important that the coke at the top of the fire be at a bright red heat, as otherwise the bottom



of the charge melting while the top is still cold will effervesce, and there will be danger of "frothing over." The charge should be completely melted in from twenty to thirty minutes, after which it is well to leave it in the fire for at least ten to twenty minutes before taking it out, tapping, and pouring. The slag, if glassy and homogeneous both in texture and colour, may be thrown away; otherwise it must be powdered, mixed with a little more litharge and lead flux, and remelted. In the case of all ores and products containing over 100 ozs. silver or 5 ozs. gold per ton the slag should be remelted as described.

The litharge or red lead used must always be tested for silver by fusing 4 A.T. with  $\frac{1}{10}$  A.T. of flour and 1 A.T. each of sodium carbonate and borax, the resulting lead button being scorified down and cupelled. The amount of silver found is deducted from the weight of the silver bead obtained. Litharge always contains an appreciable amount of silver; red lead, however, is sometimes so pure as to show only the minutest trace from the 1 A.T. taken, hence its use is preferable.

In Colorado, crucible assays for silver and gold are often made in the muffle, the quantity taken being then  $\frac{1}{2}$  A.T., as a larger quantity could not well be fluxed without using a crucible too large for the muffle. A good list of crucible charges on the muffle system is given by Furman.\*

Copper ores and furnace products are best treated by the special method which will be described for the assay of silver in copper matte.

*Cupellation.*—Cupels are ordinarily made of bone ash, ground to pass a 40- to 60-mesh sieve, and moistened with water, to which a little  $K_2CO_3$  is sometimes added. They must be very slowly and carefully dried before using, and should weigh a little more than the lead buttons which they are to receive. They must be brought to a bright red heat in the closed muffle before charging the lead buttons, and immediately this operation is completed the door should be again closed, and not re-opened until the surface of the melted buttons is seen to become bright. The temperature is then regulated so that the fumes rise lazily to the muffle arch, a glowing piece of charcoal being sometimes placed in the mouth of the muffle should it appear to be too cool in front. If the temperature be too low there will be risk of the button "freezing," in which case the assay is unreliable, and must be repeated; while, if too high, the volatilisation loss is much increased. The best rule is probably that universally accepted among American assayers, viz., to leave a distinct ring of feathery litharge crystals round the edge of the cupel, marking the area covered by the original lead button. Beads of silver cupelled at this heat always retain slight traces of lead, but the *plus* error thus introduced is much less than the *minus* loss by volatilisation (not to speak of the much greater loss by cupel absorption), so that a rough approximation to accuracy is obtained.

The actual melting point of litharge is about 950° C., but cupellation can be started at a temperature of about 800° C.; and, after starting, the temperature can be reduced to as low as from 700° to 750°, with a much lower volatilisation loss.

Just before the end of the operation the button exhibits a bright play of colours, caused by the formation of very thin films of litharge, and at this point the door should be closed completely, or the cupel should be pushed

\* *Manual of Practical Assaying*, N.Y., 1893, p. 139.

back into the hottest part of the muffle and left there for a minute or two to expel the last traces of lead. It is then withdrawn gradually towards the front of the muffle to prevent loss by "spitting" or "sprouting" at the moment of solidification, and, if at all large, is best covered with another hot cupel upside down, so as to cool it as slowly as possible.

The bead is removed from the cupel when cold by sharp-nosed pliers, and squeezed and brushed to remove adhering bone ash, if large enough; if too small, it is carefully picked up by fine forceps, placed on a blowpipe anvil and flattened by a blow from the light hammer, after which it is weighed on the assay balance to within  $\frac{1}{100}$  mg. Very minute beads are weighed on the gold assay balance, which is sensitive to within  $\frac{1}{100}$  mg., corresponding to  $\frac{1}{10}$  of an oz. gold on  $\frac{1}{10}$  A.T. taken.

During recent years substitutes for bone ash containing magnesia in various forms have come into general use, one being a mixture of magnesia and magnesium borate. All the materials for the patent cupels made from these bone ash substitutes are ground finer than ordinary bone ash, and the cupels are formed under a higher pressure (generally hydraulic), so that they are harder, heavier, and less porous, which tends to reduce the loss by absorption. Furthermore, the heat conductivity and the specific heat being both higher, and the mass of the cupel greater for a given bulk, the heat remains much more uniform during the process, so that although the muffle has to be raised to a slightly higher temperature before starting, the heat generated by the oxidation does not raise the temperature of the button so high, and consequently the cupellation throughout can be conducted at a more uniform and slightly lower degree of heat than with bone ash. For the same reason cooling takes place more slowly, and there is much less tendency to spitting. The great advantage of the patent cupel, however, is the much lower absorption loss.

*Parting.*—The bead should be flattened before parting, and, if it does not already contain at least two and a-half times as much silver as gold, a small fragment of pure silver foil is added and fused together with the bead on a cupel by means of the blowpipe flame.

Parting may be done either in a small  $\frac{3}{8}$ -inch test-tube, or in a porcelain crucible. The former method is preferable when the bead is large, owing to the danger of loss by projection when boiling a quantity of acid in a crucible; but small beads are best parted in a crucible, because of the danger that particles of gold may stick to the sides of the test-tube and escape observation, and because only one acid (of 1.20 sp. gr.) is required for small beads. Practical experience shows that on the average more gold is obtained when parting is conducted in a crucible, perhaps because the nitrous acid formed escapes more freely, so that there is less danger of dissolving gold. Large beads are parted by boiling, first in acid of 1.16 specific gravity till apparently no more action takes place; the acid is then carefully decanted off and replaced by acid of 1.27 specific gravity, with which boiling is continued for five minutes longer. The acid is then poured off and the gold washed twice with hot water, being allowed to settle *very thoroughly* \* each time before decantation, after which the tube filled with water is inverted over a porcelain crucible, which is then also filled with water, and, after tapping the tube a few times, it is allowed to stand for a minute or two. When no more

\* Stress must be laid upon this; v. Ledoux, *Trans. A.I.M.E.*, vol. xxiv., p. 873.

particles can be discerned settling through the water, the tube is removed and the water poured off, the last drops being drawn up by a capillary glass tube, and the crucible dried on a hot plate. When quite dry the gold is ignited over a spirit lamp, or in the muffle, and weighed on the "gold balance" to .01 mg., its weight being subtracted from that of the combined gold and silver bead obtained by cupellation to get at the silver.

Small beads may be advantageously parted with strong sulphuric acid instead of with nitric. The operation is much slower, but there is less risk of dissolving traces of gold, and the beads do not break up so badly.

*Assay of Slags for Gold and Silver.* At custom smelting works determinations of the silver in slag samples is usually done twice a day for each furnace, the day and the night samples being done separately, and the quantity taken for each assay being from 20 grammes to 1 A.T. A good mixture is that given in column 9 of Table I.VI., the quantities of flux there mentioned as sufficient for 2 tons of slag being reduced by one half if only 20 grammes or 1 A.T. be operated upon, as is generally the case.

For determining gold the silver buttons are allowed to accumulate for periods of from a week to a month, and are then counted and parted in the usual way. As each button represents so much weight of slag, the total quantity of slag taken corresponding to the weight of gold obtained is easily seen. Thus, to quote an actual example:

Took 400 buttons, each obtained from 20 grammes of slag, equals a total of 8 kilos. slag. Weighed 269.42 mgr., equals an average of 26.18 grammes Ag per metric ton, or 0.76 oz. per short ton. Parted in a crucible gave a residue of 0.72 mg. gold, equals 0.009 grammes gold per metric ton = .00261 oz. Troy per short ton of slag, say  $1\frac{1}{4}$  grains per ton, or in cash value, say,  $2\frac{1}{2}$ d.

*Assay of Matte and other Copper Substances for Silver and Gold.* At many smelting works the so-called "all fire" method is followed. The sample is scorified down with large excess of lead in ten lots of  $\frac{1}{10}$  A.T. each, collecting the buttons in pairs at the end, and assaying separately the cupels and slag of each pair. This method gives the best results for gold, but higher results for silver are usually given by the so-called "wet" or "combined method," which has been in use for many years at copper works where silver assays have to be made on rich copper ores and furnace products. Its most modern form is as follows:

Take from 1 to 2 A.T. of pulverised matte or copper borings, dissolve *in the cold* in nitric acid free from chlorine and nitrous acid, using 100 c.c. of acid for each A.T., diluted till just strong enough to attack the material (for copper borings 1 to  $3\frac{1}{2}$ , for matte 1 to  $2\frac{1}{2}$ ). The beaker should be kept in cold water, so as to keep down the temperature until solution is nearly complete, then heated to complete solution, adding 20 c.c. of strong  $\text{HNO}_3$ , and passing a current of air to eliminate nitrous acid as formed. Dilute to about 1 litre, add 20 c.c. of a saturated solution of lead acetate, then 1 gramme  $\text{NaCl}$  dissolved in water; stir or shake well, then add 2 c.c. of strong  $\text{H}_2\text{SO}_4$ , or, better, 5 grammes of  $\text{Na}_2\text{SO}_4$  dissolved in water; mix well, and allow to stand twelve hours. Decant off the clear liquid, filter off the precipitated  $\text{PbSO}_4$  which will have carried down with it all the  $\text{AgCl}$  and free gold in suspension, and wash it well with hot water, picking out any globules of

sulphur with a forceps. Burn these in a porcelain crucible, add the residue to the precipitate, carefully dry, and separate as completely as possible from the paper, which should be burnt at a low temperature either in a small scorifier or crucible. The ashes, together with the bulk of the dried precipitate, are mixed in a small glass or porcelain mortar with either litharge or lead flux, according as the scorifier or the crucible is employed for reduction. If the former, a cover of borax is used, and the mixture melted and poured as usual. If the latter, the mixture is charged into a small crucible with a borax cover and an iron nail, and fused in the muffle like a lead assay. If the button is over 10 grammes in weight, or if hard from the presence of copper, it is scorified down (lead being added in the latter case), and the resulting button is cupelled and parted in the usual way. The hard part of the cupel, together with the slag from the fusion of the precipitate and the scorifier slag, if any, are pulverised, mixed with 2 parts lead flux, 2 parts borax glass, and  $\frac{1}{2}$  part litharge, and fused down in the same crucible. The button of lead is cupelled, and the silver obtained added to the original assay.

Whitehead\* recommends separating the gold before precipitating the silver, and using KBr instead of NaCl for this purpose, claiming that by the use of NaCl there is danger of dissolving gold through liberation of free chlorine. Experiments by Van Liew,† however, have proved that if the nitric acid is sufficiently dilute and the excess of NaCl only slight there is no risk of dissolving part of the gold. Whitehead's recommended method also requires two additions of lead acetate and sulphuric acid, two separate settlings of twelve hours each, and two fusions and cupellations, one for the gold, and one for the silver. It is much more convenient to get both Au and Ag together in a single button after only twelve hours' settling, and this is the common practice; the use of KBr (or KI), however, is a decided improvement, as it obviates the danger of losing gold through Cl set free, and gives somewhat higher results for Ag owing to the less solubility of AgBr (AgI) than of AgCl in solutions of copper nitrate. It is best to perform the assay in duplicate from the beginning so as to get a check on both silver and gold.

**Losses of Silver in the Commercial Assay.**—Whenever accurate results are desired, it is necessary to take the whole of the slags produced in the assay (whether from crucible or scorifier, or both) together with the hard portion of the cupel, and, after powdering, make a separate assay of the mixture as of a basic lead ore, adding some fluorspar to assist in melting the refractory bone ash. The lead button is cupelled at a low temperature, and the silver bead obtained is added to the original assay. Where extreme accuracy is desired, it is even necessary to repeat the correction. The assays made at metallurgical works for ascertaining the value of purchased ores, except those of very high value (and generally for all other purposes), are almost invariably *direct uncorrected* assays; hence, such establishments have a considerable margin to go upon, and the working results obtained appear to be more satisfactory than they really are.

Some interesting researches,‡ however, show that the commercial assay

\* *Chem. News*, 1892, vol. lxvi, p. 19.

† *E. and M. J.*, April 28, 1900, p. 499.

‡ Dewey, *Journ. Amer. Chem. Soc.*, 1894, vol. xvi, pp. 505 to 516; Stetefeldt, *Trans. A.I.M.E.*, vol. xxiv, pp. 530 to 538. Furman, *ibid.*, pp. 735 to 742; Oemichen, *Zeitschr. f. angewandte Chemie*, 1893, p. 723.

for silver is even more inaccurate than has been supposed. For details the student should consult the original papers, but it may be briefly stated that of the three sources of error, viz.:—(1) Loss from absorption by the cupel, (2) loss in the slags, and (3) loss by volatilisation—it is clearly shown by Dewey that the first is generally by far the most important, the others following in the above order. Some of his results are given in Table LVII., compared with similar results by Stetefeldt.\*

TABLE LVII.

Material.	Value per ton ounces.	DEWEY.		STETEFELDT.	
		Losses in Slag and Cupel.		Total Losses.	
		Range of Losses.	Average.	Range of Losses.	
Tailings, . . . . .	1 to 20	18.0 8.6	13.7	21.8 8.4	
Washed ore, . . . . .	15 „ 35	13.0 10.0	10.4	19.7 5.0	
Raw ore, . . . . .	20 „ 50	16.0 5.0	9.7	16.8 9.0	
Roasted ore, . . . . .	20 „ 45	12.0 6.0	9.1	10.6 5.4	
Lead carbonates, . . . . .	50 „ 100	5.1 2.6	3.8	4.8 2.9	
Base sulphides, . . . . .	1,000 „ 5,500	5.0 1.8	3.0	5.1 2.2	
“Regular” sulphides, . . . . .	2,000 „ 16,000	2.8 1.2	1.7	4.9 1.2	
Copper-silver bullion, . . . . .	9,000 „	2.0	2.0	„	
Fine bullion (999.3 fine), . . . . .	20,100 „	1.1	1.1	„	

The loss of gold in slag and cupel is much less than that of silver, as shown by the experiments of Miller and Fulton,† but in practice the differences in commercial gold assays are much larger in proportion, owing, no doubt, to errors of manipulation and to the smaller quantity of metal operated upon. Furman‡ found losses in cupelling amounting to from  $1\frac{1}{2}$  to 7 per cent. of the silver, and 0 to 12 per cent. of the gold; he places the average loss of silver at 2.58 per cent., which agrees fairly well with the results of Dewey.

On *lead bullion* Oemichen§ gives cupellation losses varying from 5 to 10 per cent. of the silver, and from 2 to 5 per cent. of the gold; Hofman|| gives the loss on a pure 41 ozs. bullion as 3.5 per cent. of the silver. In the cupellation of argentiferous copper and lead, Rösler¶, found losses as in Table LVIII.

\* *Loc. cit.*, p. 537.

† *S.M.Q.*, vol. xvii., Jan. 1896, p. 169.

‡ *Trans. A.I.M.E.*, vol. xxiv., p. 735; also *E. and M. J.*, Nov. 3, 1894.

§ *Zeitschr. f. angewandte Chemie*, 1893, p. 723.

|| *Trans. A.I.M.E.*, vol. xxiv., p. 868.

¶ Quoted by Hofman, *Metallurgy of Lead*, 1893, p. 249.

TABLE LVIII.

Material.	Ozs. Ag. per ton.	Charge Employed.		Ratio of Ag to Pb 1 :	Percentage Loss of Silver.
		Mgrms. Ag.	Grms. Pb.		
Copper, . . . . .	43·75	10+10 Cu	200	12,000	8·3
Copper, . . . . .	583·32	200+10 Cu	160	800	4·5
Lead, . . . . .	43·75	150	100	600	2·5
Do., . . . . .	87·49	150	50	300	2·2
Do., . . . . .	218·74	150	20	120	2·0
Do., . . . . .	437·49	150	10	60	1·6
Do., . . . . .	874·98	150	5	30	0·9
Do., . . . . .	4,374·90	150	1	6	0·4

Ledoux\* found losses of 1·75 per cent. in assaying copper bars with 92 ozs. of silver, and of 1·12 per cent. in the case of matte with 182 ozs. of silver. In direct scorification of black and blister copper containing from 70 to 80 ozs. per ton with test lead the author has found uncorrected results to range from 5 to 8 per cent. lower than the assay by the combined wet and dry method above described, the chief source of loss in this case being no doubt the large amount of scorifier slag produced.

Hook† gives the following table for losses of silver as determined by the recoveries from slags and cupels:—

TABLE LIX.

Silver in Ore Charge, Milligrammes.	Loss per cent.	Silver in Ore Charge, Milligrammes.	Loss per cent.
33·1	2·23	177·9	1·29
54·6	1·86	518·9	0·86
97·7	1·61	939·9	0·72
148·6	1·48	1648·9	0·70

It is to be supposed that in this case there was no large amount of copper or tellurium present.

The experiments of Kauffman‡ show that not only is the cupel-absorption large in proportion as the quantity of silver present is small, but it also increases with increased weight of lead buttons to be cupelled away. Other data in regard to cupel losses may be quoted as follows:—

Beringer's experiments, . . . . .	Loss of 1·62 per cent.
Mason and Bowman, . . . . .	„ 1·99 „
H. R. Edmunds', § . . . . .	„ 2·18 „

The last-named author finds that the depth of the cupel appears to have an important effect on the speed of cupellation, shallow cupels being 20 per cent. quicker than deep ones, but it does not follow that the cupel absorption is necessarily greater, although that probably is the case.

\* *Trans. A.I.M.E.*, vol. xxiv., p. 875.

† *E. and M. J.*, June 14, 1902, p. 829.

‡ *E. and M. J.*, March 28, 1908, p. 661.

§ *E. and M. J.*, Aug. 12, 1905, p. 245.

It should be noted that the percentage losses through cupel absorption above quoted have all been determined when employing quantities of silver largely in excess of those commonly obtained as the result of ordinary assays. Seeing that the cupel absorption does not depend entirely upon the quantity of silver present, but to a much greater extent upon the quantity of lead to be cupelled away, it seems probable that upon any such quantities of silver as are ordinarily handled in the form of assay buttons from ore assays the losses range from 5 to 6 per cent. on the quantity present.

#### ANALYSES OF ORES AND SLAGS.\*

The methods employed in smelting works for the analyses of ores and slags must be fairly accurate and, at the same time, rapid, for otherwise it becomes impossible to get analyses done within twenty-four hours after taking the sample, as should always be the case. Particularly as regards slags is rapidity of vital importance, for the removal of some trouble connected with the working of the furnace frequently depends on ascertaining the composition of the slag being produced, and in this case a slag analysis to be of any use should be completed within a couple of hours or so. In order to secure such rapidity it is important to be able to decompose the slag with acid quickly and completely without a preliminary fusion. This can be always done, provided the slag sample is in a vitreous condition and very finely pulverised. Slag samples should be always taken by inserting an iron rod or small ladle into the potful of molten slag several inches deep, removing, and instantly plunging into a deep vessel of cold water, by which means the silicates are prevented from crystallising and remain in the unstable vitreous condition. The final sample of 30 to 50 grammes should always pass a 120-mesh sieve, and the portion of a few grammes which is to be weighed out for analysis should be finely ground in an agate mortar. If this be done all lead slags and most lead and silver ores will be completely decomposed by boiling in HCl for a few minutes. Another essential condition of rapidity is the use of the smallest possible quantity of water and reagents.

The case of an easily decomposed ore or slag will first be considered :—

*First Portion for FeO.*—Weigh  $\frac{1}{2}$  gramme of slag finely ground in agate mortar, mix in porcelain casserole with 20 c.c. boiling water, add 15 c.c. strong HCl, stir well, boil till all action ceases, then adopt one of the following procedures :—

1. Add 5 drops saturated  $\text{SnCl}_2$  solution, allow to cool; when cold add 15 c.c. of  $\text{HgCl}_2$  solution to peroxidise any excess of tin, and titrate with  $\text{K}_2\text{Cr}_2\text{O}_7$ , using potassium ferricyanide as an indicator.

2. Add granulated zinc in excess to reduce the iron, run the solution through a small filter, add a few c.c. of  $\text{H}_2\text{SO}_4$ , dilute, and titrate with permanganate solution.

The second method is considered more convenient, but is less accurate unless the same quantity of HCl be used in standardising the permanganate, when it gives results which are sufficiently accurate for smelter purposes.

This method of dissolving slags in HCl without oxidising agent has the advantage of showing at once if there is any ferric oxide in the slag by the

\* v. Furman, *Manual of Practical Assaying*, N.Y., 1893, p. 79<sup>1</sup>; also Argall, *Western Mill and Smelter Methods of Analysis*, Denver, 1905.

yellow colour produced in the solution in that case; when it occurs, this is usually an indication of insufficient reduction, and is, therefore, important.

*Second Portion for Silica (Insoluble) plus BaSO<sub>4</sub> and CaO.*—Weigh  $\frac{1}{2}$  gramme ground in agate mortar as above, mix in porcelain casserole with 2 c.c. water and a pinch of KClO<sub>3</sub>; add 5 c.c. strong HCl, and stir well, then add 3 c.c. strong HNO<sub>3</sub>, boil till all action apparently ceases, and add a few drops of H<sub>2</sub>SO<sub>4</sub>. Evaporate to dryness, ignite gently, cool, boil out with 50 c.c. of water and 10 drops of H<sub>2</sub>SO<sub>4</sub>, filter, and wash several times with hot water. Digest the residue with ammonium acetate several times (this filtrate and washings should be saved separately, and will serve for a check wet assay for lead by the molybdate method already described). Wash the residue with two drops of ammonia water, and again with hot water, dry, ignite, and weigh SiO<sub>2</sub> + BaSO<sub>4</sub>. If much of the latter be present, it must be determined by fusion with alkaline carbonate as described below, or the ignited precipitate may be evaporated down twice with HF to eliminate SiO<sub>2</sub>, any residue being BaSO<sub>4</sub> and insoluble silicates.

Boil the filtrate, make alkaline with AmHO, add 5 c.c. AmHO in excess (to ensure the formation of sufficient oxalate to keep MgO in solution), then add a saturated solution of oxalic acid till the precipitate of ferric hydrate just redissolves on boiling. Boil for a minute or two, filter, and wash the precipitate on the filter several times with boiling water. Pierce the filter, wash the oxalate of lime into a 4-oz. flask with a jet of boiling water, pour 5 c.c. of dilute H<sub>2</sub>SO<sub>4</sub> over the filter, and wash it all down into the flask, heat to boiling, and titrate immediately with standard permanganate solution, the standard for CaO being one-half of that for Fe.

A sample of readily decomposed ore or a slag can be easily analysed roughly for SiO<sub>2</sub>, BaSO<sub>4</sub>, FeO, and CaO in from one and a-half to two hours.

According to Ulke,\* lime can be separately determined in a slag sample in twenty minutes by the following method:—Dissolve  $\frac{1}{2}$  gramme in 30 c.c. hot water and 6 c.c. nitric acid, agitating steadily to prevent adherence, dilute to 150 c.c., add 1 drop H<sub>2</sub>SO<sub>4</sub> to precipitate any barium, and boil. Add ammonia drop by drop to approximate neutralisation, the solution becoming deep red, but no iron being actually precipitated. Add 3 grammes ammonium oxalate, cover, boil for three minutes, filter through a No. 597 filter, wash with hot water, then once with ammonia water and twice with hot water, then once with ammonia water and twice more with hot water, and titrate as usual with permanganate.

*Ores containing Insoluble Silicates and much BaSO<sub>4</sub>.*—Take 1 gramme ore or  $\frac{1}{2}$  gramme slag, decompose with HCl, HNO<sub>3</sub>, and H<sub>2</sub>SO<sub>4</sub>, evaporate to dryness and digest the residue with ammonium acetate to dissolve out lead sulphate, exactly as in the first method, after filtering off the soluble sulphates. Dry the residue and fuse either in a platinum crucible with Na<sub>2</sub>CO<sub>3</sub>, or in a large silver crucible with KHO for thirty minutes, disintegrate the fused mass in water, boil, filter, and wash well. The filtrate containing alkaline silicate may be evaporated to dryness twice with HCl and the residual SiO<sub>2</sub> filtered off, dried, ignited, and weighed; or, if the silica has been determined on a separate portion, this filtrate may be thrown away. Dissolve out the residue in 3 c.c. HCl and 10 c.c. water, boil, reprecipitate with a few drops of H<sub>2</sub>SO<sub>4</sub>, filter, wash, dry, ignite, and weigh the BaSO<sub>4</sub>. The HCl filtrate from the

\* *E. and M. J.*, Feb. 10, 1900; *Min. Ind.*, 1900, vol. ix., p. 442.



$\text{BaSO}_4$  contains the  $\text{Al}_2\text{O}_3$ ,  $\text{CaO}$ , and other bases of the insoluble silicates; it may either be added to the first acid filtrate from the decomposition of the ore or the bases may be determined in it separately.

*Alumina* and *magnesia* may be determined in any of the foregoing portions by ordinary gravimetric methods.

*Zinc*.—Dissolve a separate portion of 1 gramme, remove the  $\text{SiO}_2$  and Pb as above, peroxidise with  $\text{KClO}_3$ , add large excess of  $\text{AmCl}$  and  $\text{AmHO}$ , filter at once, and titrate the zinc in the cold filtrate slightly acidified with  $\text{HCl}$  by means of standard ferrocyanide solution, using uranium acetate as the indicator and pure  $\text{ZnO}$  for standardisation. With ores containing much iron the results are always low, owing to the obstinate retention of Zn by the iron precipitate, and for anything like accurate results the hydrate precipitate should be redissolved in a considerable quantity of  $\text{HCl}$  and reprecipitated, using large excess of ammonia. The separation of iron as basic acetate is more complete, but in that case the zinc cannot be directly titrated in the filtrate.

A rapid and satisfactory way of determining zinc and lead in a single portion is the following :—

Dissolve  $\frac{1}{2}$  gramme ore in 10 c.c. of 1 to 3  $\text{H}_2\text{SO}_4$  and 5 c.c. strong  $\text{HNO}_3$ , boil down in a porcelain capsule till strong fumes of  $\text{SO}_3$  are given off, cool, dilute to about 40 c.c., filter off the  $\text{PbSO}_4$  and insoluble matter, from which the former is dissolved by am. acetate and titrated as usual with molybdate. To the filtrate add 200 grains potassium bitartrate, a *slight* excess of  $\text{AmHO}$ , but only to feeble alkalinity, and dilute to about 250 c.c., adding also a little  $\text{Fe}_2\text{Cl}_6$  (unless the ore contains 10 per cent. of iron or upwards). Titrate with ferrocyanide, using as indicator glacial acetic acid spotted on a plate, the end reaction, which is a sharp one, being indicated by formation of Prussian blue. The reaction differs from that which takes place in an acid solution, but appears to be quite uniform, and pure zinc oxide is used for standardisation. The standard solution is made by dissolving 46 grammes of the salt in 1 litre of water, and for standardising 12.461 grammes  $\text{ZnO}$  is dissolved in dilute  $\text{HCl}$  and made up to 1 litre. Of this solution 10 c.c. mixed with 5 grammes potassium tartrate, a few drops of  $\text{Fe}_2\text{Cl}_6$ , ammonia to feeble alkalinity, and water to 250 c.c. should take 10 c.c. of the ferrocyanide solution.

Manganiferous ores and slags should be dissolved in  $\text{HNO}_3$  alone, evaporated to dryness and taken up with  $\text{HNO}_3$  and  $\text{KClO}_3$ . Then filter off the insoluble matter, oxides of Fe and Mn, &c., through glass wool, wash with dilute  $\text{HNO}_3$ , dilute, and proceed as above.

*Manganese* is best determined in a separate portion of 1 gramme by Volhard's method, using sulphuric acid in the evaporation to separate Pb and Ba with the  $\text{SiO}_2$ , and then precipitating  $\text{Fe}_2\text{O}_3$ ,  $\text{Al}_2\text{O}_3$ , &c., from the filtrate by means of an emulsion of zinc oxide in excess in a graduated  $\frac{1}{2}$ -litre flask, the contents of which are then brought up to the mark with distilled water, and after thorough shaking allowed to settle. When the supernatant liquid is practically clear, 100 c.c. are, by means of a pipette, transferred to an 8-oz. flask, brought to a boil and titrated with standard permanganate, the standard of the solution for Mn being 0.2964 of its standard for Fe. The error introduced by the volume of the precipitate of ferric hydrate with excess of zinc oxide may be usually disregarded, but, if necessary, a correction

may be applied by weighing the precipitates from several determinations after washing, drying, and igniting, and calculating the volume of solution displaced.

*Copper* \* may be determined either by the battery, by the iodide method, or by cyanide; in either case it should be first separated from the solution by precipitation with Zn (Al) or with sodium hyposulphite in a boiling acid solution. The electrolytic and iodide methods are preferable for high-grade copper products, but, according to the author's experience, no method gives more accurate results on low-grade ores than titrating against a "check" with cyanide, after previous precipitation with pure Zn or Al, or as  $\text{Cu}_2\text{S}$  with hyposulphite.

*Arsenic* is best determined by Pearce's method.† The arsenic is precipitated as sulphide or as ferric arseniate, converted into alkaline arseniate by fusion with  $\text{KNO}_3$  and  $\text{Na}_2\text{CO}_3$ , and precipitated as  $\text{AgAsO}_4$ , which may be either reduced to Ag by a scorification assay and calculated back, or determined direct by solution in dilute  $\text{HNO}_3$  and titration with standard solution of ammonium sulphocyanate after addition of ammonio-ferric sulphate to serve as indicator.

*Sulphur* is determined by ordinary methods, weighing as  $\text{BaSO}_4$ . One method is as follows:—

Take 1 gramme, add 10 c.c. of a saturated solution of  $\text{KClO}_3$  in  $\text{HNO}_3$ , allow to act for fifteen minutes in the cold, then evaporate to dryness, take up in 10 c.c.  $\text{HCl}$  and 10 c.c. water and boil. Add 20 c.c.  $\text{AmHO}$  and 5 c.c. hydrogen peroxide, and filter off lead and iron. Well wash with hot water, acidify, filtrate with  $\text{HCl}$ , boil well, add  $\frac{1}{2}$  gramme  $\text{BaCl}_2$  in solution. Allow to settle, filter, wash, and ignite ppt. of  $\text{BaSO}_4$ .

The fies method (fusion with caustic potash in a silver dish) also gives accurate results, and is perhaps somewhat quicker.

A simple method of ascertaining roughly the amount of sulphur contained in a given ore is by determining the amount of matte formed on melting it with carbon, borax glass, and iron. Five grammes of ore are mixed with 15 grammes of borax glass and 3 grammes of charcoal, and two nails are added. The mixture is then melted down in a hot fire for fifteen minutes, taken out, and poured, after removing the nails. The button of matte produced is, when cold, carefully removed from the slag (and from the lead button, if any), and weighed. With copper-free ores it may be reckoned at 30 per cent. S, but with ores containing much copper or arsenic it is better reckoned as 25 per cent. S. In any case the method gives only a rough approximation, but it is sometimes useful in works' laboratories which are pressed for time, because it can be carried out by an unskilled helper, and the determination of S by this means, in ores containing not too much As, is generally near enough for charge-mixing purposes, as well as for checking the work done in the roasting department. The amount of matte produced in the blast furnace is, of course, always less than is indicated by the above assay.

\* v. Torrey & Eaton, *E. and M. J.*, May 9, June 6, June 27, 1885; also Low, *Proc. Colo. Sci. Soc.*, vol. i., pp. 17, 69; Peters, *Modern Copper Smelting*, 1895, pp. 41-65; Beringer, *Text-book of Assaying*, 1906, &c.

† *Proc. Colo. Sci. Soc.*, vol. i.

## CHAPTER XX.

### THE TREATMENT OF ZINC-LEAD SULPHIDES.

REFERENCE has been made in Chapters vii. and ix. to the difficulty of treating, by the ordinary roast and reduction process, such argentiferous lead ores as contain 20 per cent. and upwards of zinc, except by diluting or smothering them with an admixture of purer ores. The object of almost all attempts at treating such material has been to remove the zinc in solution, so as to smelt the plumbiferous residue more easily and with less loss.

**Concentration.**—But few zinc-lead sulphide ores contain galena and blende so intimately mixed as to put them entirely beyond the reach of ordinary dressing processes to separate a notable proportion of the galena in a condition suitable for smelting. Concentration, therefore, should usually be the first step in the treatment of such ores; unfortunately, however, not only is a large amount of “middlings” produced, which it is quite beyond the power of ordinary dressing machinery to separate, but there is a great loss of both lead and silver in fine slimes, the whole of which have, therefore, to be carefully settled and collected for smelting.

The subject of ore-dressing in general does not fall within the scope of a treatise on Metallurgy, and reference should be made to standard works on the subject,\* but the practice at Broken Hill is so peculiar and influences to such an extent the subsequent metallurgical practice in that important district as to warrant a short reference here. All the Broken Hill dressing plants have been recently much altered, and the following remarks refer to their present condition.

The plant of the *Broken Hill Proprietary Company* † consists of two separate mills, each of 6,000 tons per week capacity. The ore, having been already crushed to  $1\frac{1}{2}$  to 2-inch size in an independent crushing plant, is fed from the bins in each mill by means of a roller feeder direct into the coarse rolls (3 sets), from which it passes to shaking screens with  $\frac{3}{16}$  holes. The through size goes to the jigs, the over-size to a travelling belt discharging into an elevator, which raises it to bins over the three sets of medium rolls set at  $\frac{1}{8}$  inch and running at 45 revolutions per minute. Again the through size from the shaking screens which follow these rolls goes to the jigs, while the over-size is elevated to the three sets of fine rolls set at  $\frac{1}{16}$  inch apart and running at 78 revolutions per minute, the over-size from which goes to No. 5 Krupp ball mills for wet grinding.

The crushed ore passes through upward-current cone-separators, which eliminate slimes for the slime department, and the discharge passes to the May double-plunger 5-compartment jigs, yielding in the first two compartments 60 per cent. lead concentrates, in the third and fourth a middle product with 10 per cent. Pb, 19 per cent. Zn, and 10 ozs. of silver, and in the last

\* Amongst which special reference should be made to the great work by Richards.

† Delprat, *Trans. Aust. I.M.E.*, 1907, xii., pp. 1-29.

tailings with 3 to 4 per cent. lead. The discharge from these jigs is continuous through spigots, and the concentrates run into draining bins, whence they are loaded into railway trucks. The tailings are also drained in bins before being trammed to tailings heaps. The middle product is run into No. 4 Ball mills with  $\frac{3}{4}$ -inch slot screens, and the discharge is elevated into 4-compartment fine jigs after separation of slimes as before. The fine jigs make concentrates with Pb 40 per cent., Zn 13 per cent., and Ag 22 ozs. per ton; middlings with Pb 9 per cent., Zn 18 per cent., and Ag 10 ozs. per ton; and tailings with Pb  $5\frac{1}{2}$  per cent., Zn 18 per cent., and Ag 8 ozs. The concentrates, middlings, and tailings from these jigs are handled in the same way as those from the coarse jigs.

The "slimes" from all the slime separators pass into V-boxes or Spitzkasten separators, where they are classified into coarse and fine slimes. The coarser size goes to Wilfley tables, the middlings from the first set of which pass to a second set. The finer slimes go to two sets of Lührlig vanners. The concentrates from tables and vanners all pass automatically to concentrate bins, and the tailings to bins, from which they are removed to the zinc-treatment plant.

During the various operations of crushing and grinding no less than 12 per cent. by weight of the ore is converted into a true slime, which contains Pb 18 per cent., and Ag 17 ozs. per ton. This is separately collected, and its treatment has been described in Chapter v.

The recovery of lead at this plant is as follows:—Coarse jigs 47·5 per cent., fine ditto 13·5 per cent., Wilfley's 3·15 per cent., Lührlig vanners 3·15 per cent.—total, 67·3 per cent. The quantities treated per hour per machine are as follows:—Double coarse jigs  $5\frac{3}{4}$  tons, double fine jigs  $2\frac{3}{4}$  tons, Wilfley tables  $\frac{3}{4}$  ton, single belt Lührlig vanners  $\frac{1}{2}$  ton. The average composition of the final products is as follows:—

	Per cent. Pb.	Per cent. Zn.	Ozs. Ag. per ton.
Lead concentrates (average), . . . . .	55	10	26
Jig tailings, . . . . .	4	12	5
Table and vanner tailings, . . . . .	5	17	8
Slimes, . . . . .	18	17	17

The plant of the *B. H. Block 10 Co.\** consists of four independent units, besides two auxiliary units for retreating the tailings from the four main units whenever the price of metals is high enough to show a sufficient margin over the additional working costs. The ore is crushed at the shaft to  $1\frac{1}{2}$ -inch size, and carried to the mill by an aerial ropeway. Each unit comprises, first, a shaking screen with 3 mm. punched holes fed by rollers, the over-size from which gives to a set of rolls 30 inches in diameter by 15 inches face run at 14 revolutions per minute, the maximum capacity of which is 1,000 tons per week. The crushed ore is run through two trommels, 6 feet long with  $\frac{3}{4}$ -inch holes, and the over-size is returned to the same rolls, experiments with two sets of rolls in tandem not having reduced the production of slime sufficiently to justify the extra power cost.

\* Williams, *E. and M. J.*, May 8, 1909, p. 939.

The through size from the trommels passes over a small conical separator to eliminate slimes, and the discharge passes to a double-compartment coarse jig, the first two compartments of which yield concentrates, while the third and fourth yield middlings, which are elevated to a No. 4 Krupp ball mill that grinds 4 or 5 tons per hour to pass a  $\frac{3}{8}$ "-hole. The reground pulp passes to a small cone separator, sending a discharge to a double-compartment fine jig and an overflow to a Wilfley table. The tailings and slimes from this table, together with the overflow from the cone separator, go to slime thickeners.

The fine jig produces concentrates from the first two compartments, middlings from the third and fourth compartments, which are elevated and returned to the Krupp regrinding mill, and tailings which are elevated to two 5-foot Forwood-Down grinding pans. The product from these is concentrated on two Card tables, yielding concentrates, tailings to waste, and middlings which go to a Spitzkasten A; the discharge from this goes to two single-belt slimes' vanners that also yield concentrates to bins, tailings to waste, and middlings, the latter going to the "vanner extension" plant.

The slimes' plant receives fine sands and slimes from the "thickeners," and is common to the whole four units; the overflow from the thickeners, after passing through a sump which receives all wash and clear overflow waters, is sent through V settlers and returned to the rolls. The discharge from the thickeners is still further thickened in four Spitzkasten, fed equally from a distributing box, and the charge is distributed to ten Card tables, yielding concentrates to bins, tailings to V settlers, and middlings; those from six tables going to Spitzkasten A, and those from the other four tables, after passing a single Card table, to the "vanner extension" plant. All slimes from the V settlers are collected in a sump, and pumped to a "retreatment plant," comprising four Wilfleys and five vanners. The middlings from the "retreatment plant" go to the "vanner extension" plant, comprising a series of V settlers, four Spitzkasten, nine single-belt vanners, and one card table.

Strict check is kept upon the weight and assay value of the products from the two auxiliary plants ("retreatment" and "vanner extension") in order to be able to suspend operations in them immediately the price of lead falls off to a point which renders them unremunerative.

The following figures show the proportionate quantities and assay values of the products of the mill for the half-year ending March 31, 1908, during which period 67,054 long tons of ore were put through :—

Product.	Percentage by Weight.	Lead, Per cent.	Zinc, Per cent.	Silver, Ozs. per ton.
Concentrates, . . . . .	17.5	61.2	9.4	32.4
Jig tails, . . . . .	28.9	3.7	15.4	6.7
Vanner tails, . . . . .	10.7	6.2	24.2	11.0
Jig middlings, . . . . .	2.0	6.6	23.6	10.5
Wilfley tails, . . . . .	1.6	5.4	18.8	9.1
Wilfley extension tails, . . . . .	2.5	5.0	22.6	9.4
Reground tails, . . . . .	22.3	3.6	23.3	8.1
Slimes, . . . . .	14.5	10.2	24.2	14.6
Original ore, . . . . .	100.0	15.1	18.7	13.7

The recovery in concentrates was 72·1 per cent. lead, 46·6 per cent. silver; the total cost of dressing 6s. 3d. per long ton.

**Treatment of Mixed Ores and Middlings Low in Lead.**—Upon simple inspection of the assay composition of the products from the Broken Hill mills, it appears that the jig tailings, which contain about 4 per cent. Pb and 12 to 15 per cent. Zn as sulphides, the remainder being gangue, should be amenable to further mechanical concentration, and it is in fact these which are chiefly the object of the flotation processes described in the following chapter. The jig middlings and vanner and table tailings, which contain 5 to 6 per cent. lead and 17 to 24 per cent. zinc with 8 to 11 ozs. of silver per ton, are similar in composition to a variety of natural ores, which also are not amenable to a preliminary concentration for partial recovery of lead.

The number of processes suggested for treating such ores, and separating the zinc from the lead contents, is large indeed,\* but only those need be mentioned here which have shown some approach to commercial practicability. They may be conveniently grouped under the following heads, viz. :—

- A. Volatilisation and other dry methods.
- B. Solution and precipitation methods.
- C. Electrolytic methods.

Flotation processes chiefly find their field in separating the valuable sulphides in such mixed ores from the gangue, and will be described in a separate chapter.

#### A. VOLATILISATION AND OTHER DRY METHODS.

1. **Distillation in Retorts.**—The roasted ore, mixed with anthracite as reducing agent, is charged into vertical or inclined retorts exactly as in ordinary zinc smelting, and distilled upwards, leaving lead and other metals in the residue. This method is adopted on quite a considerable scale at various places in Germany and Belgium, but the retorts are badly attacked by the lead, and the method is too costly in this respect, as well as in regard to labour and fuel for adoption at most smelting centres, such as those of the United States, Mexico, and Australia.

The *Sulman and Picard* † distillation process can be carried out upon mixed lead-zinc ore carrying so high a percentage of lead as to be almost impossible to work in the ordinary way, owing to slagging the retorts.

The well-roasted ore is mixed with about 20 per cent. by weight of crushed coking coal, and the mixture is briquetted, using 5 per cent. pitch as a binder. The briquettes are of suitable shape to fit the retorts, each of which takes fifteen briquettes. Distillation is conducted in the ordinary way, though at a somewhat lower temperature, and takes no more than the normal time. In consequence of the lower temperature, but little lead is volatilised, and the zinc is of unusually high purity (viz., 99 per cent. or over); the lead, of course, is reduced to metal, but is completely enclosed in the pores of the coherent mass of coke formed by the destructive distillation of the coal

\* See the successive volumes of the *Mineral Industry*, under the head of "Zinc," where short descriptions will be found of most of the methods suggested.

† *Trans. Inst. Min. Met.*, vol. x., 1901-02, p. 432.

and of the pitch used as binder, so that the retorts are not affected. The total cost of briquetting—viz., 3s. 10d. per ton—is an addition to the ordinary cost of zinc smelting; against this, however, may be set some saving of time in charging, some additional price obtainable for the purer product, and the advantageous condition of the residues for subsequent treatment by smelting.

**2. Distillation in Blast Furnaces.** Many investigators have attempted to utilise commercially the blast furnace as an appliance for distilling zinc, but the great difficulty in the way of all such attempts is the fact that at a high temperature the reverse reaction



comes into play, even when the proportion of CO in the current of gases is very much less than that of CO<sub>2</sub> in fact, according to Lencavech,\* when it is only 0.5 per cent. of the gas employed. *Hempel's* plan† is to prepare and reduce the charge in a closed retort, in which 1 part roasted zinc, 3 parts of coal, and 0.05 part of quicklime are heated, and subsequently allowed to cool without admission of air. The coked mass is then charged into a blast furnace with closed throat, the gases from which are drawn off by means of a centrifugal fan at 1,000 to 3,000 revolutions per minute, passing on their way through a long iron pipe, in which they cool down to 30° C. The zinc fume in the cooled gases is by the centrifugal force driven against the fan casing, and so agglomerated and compacted that it falls off into a closed hopper below as a fairly dense zinc dust, which can be drawn from the hopper into bags, while the particles too fine to settle are caught in a bag house. The fume is stated to contain 72 to 90 per cent. Zn, and by means of a briquetting machine can be compressed with a small quantity of finely divided carbon into blocks suitable for retorting, and so yielding a very pure spelter; while the residue with 41.6 per cent. of SiO<sub>2</sub>, 2.93 per cent. FeO, 0.6 per cent. CaO, 33.6 per cent. Zn, 8.1 per cent. S, and 1.05 per cent. PbS, is returned to the blast furnace. The process does not appear to have been carried out on a commercial scale.

*Lungwitz*‡ proposed to conduct the smelting operation in the blast furnace under a pressure of 100 lbs. per square inch, in order to raise the boiling point of zinc above the temperature of the hearth, and so prevent the reduced zinc from boiling off as fast as formed. A furnace on this principle 3 feet in diameter at the tuyeres, 5 feet 5 inches at the bosh, and 30 feet high, was erected at Warren (N.H.) in 1905. The zinc vapour, which condenses in the upper and cooler portion of the furnace, percolates down in droplets to the hearth, accompanied by the lead present in the ore, which seems to have an important effect in retarding the re-vapourisation of the zinc, so that an alloy of lead and zinc collects in the hearth, being protected somewhat from the high temperature of the smelting zone by the layer of slag above it, and can be tapped off at intervals. The subsequent separation of lead from zinc is conducted by ordinary methods (cooling and drossing as regards the lead; redistillation as regards the zinc). The method, however, has not yet proved commercially successful.

\* *Mem. Soc. Ingen. Civils*, 1877, p. 508.

† *Berg. u. Hüttenm. Zeitg.*, 1893, Nos. 41 and 42.

‡ *E. and M. J.*, April 28, 1906.

3. **The Bartlett Process.\***—This process, in successful work for many years at Cañon City, Colorado, aims at volatilising all the lead and part of the zinc contents of mixed sulphides, which are marketed in the form of a pigment, and then smelting the residues in order to concentrate their copper and precious metal contents in the form of a matte. The ores treated by the process are those which, on account of their high zinc contents, are penalised by ordinary lead smelters, and comparatively low in silver, because, the object being to volatilise lead as well as zinc, a considerable percentage of the silver contents also is volatilised with those metals, and so lost. The ores which contain upwards of 20 per cent. zinc go through a sintering or "blowing-up" process before being smelted; such as contain less than 20 per cent. zinc are smelted direct in the blast furnace, together with the sintered residues from the other furnace.

The *blowing-up furnace* is shown in Fig. 303, in which A is the ashpit, B the troughed and perforated grate, 6 feet long by 3 feet 6 inches wide, C and D the ashpit- and working-doors respectively, E the charge-pockets, F the flue, and G a row of hollow cast-iron perforated columns, which serve to support the roof, while acting as tuyeres and supplying heated air above the grate for oxidation of the volatile matters.

The ore is crushed through a screen with  $\frac{1}{2}$ -inch mesh, mixed with 15 to 20 per cent. of moistened slack coal, and charged into the pockets, in the lower part of which above the columns it becomes heated and the coal becomes coked. The clinker from the charge having been removed through the working door, the new charge which runs down from the pockets between the perforated columns is spread over the grate to an even depth of about 6 inches, the blast is turned on, and a new charge is thrown into the pockets to warm up. The blast enters below the grate under a pressure of 4 to 8 ozs., and passes both through the grate and above it through the perforations in the hollow columns, ensuring perfect oxidation of the volatilised fumes. Each charge weighs about 500 lbs., and is completely agglomerated, and has its zinc contents scorified or volatilised in from twenty to forty minutes, the average being thirty minutes, and the total output of each furnace 6 tons per twenty-four hours. Complete expulsion of the sulphur is by no means aimed at, since sufficient must be always left to form a matte in the subsequent smelting operation, and to protect the silver from volatilisation, which is considerable, in presence of much As, Sb, Se, or Te; but comparatively little in the absence of these elements. The roasted ore is generally converted into a solid clinker, any portions not completely agglomerated being returned to the next charge. Its composition varies greatly, as shown by the following figures:— $\text{SiO}_2$  (insoluble)  $27\frac{1}{2}$  to  $30\frac{1}{2}$  per cent.,  $\text{FeO}$  36 to 50 per cent., Zn 10 to 19 per cent., S  $5\frac{1}{2}$  to  $10\frac{1}{2}$  per cent., Cu 0 to 2 per cent., C about 3 per

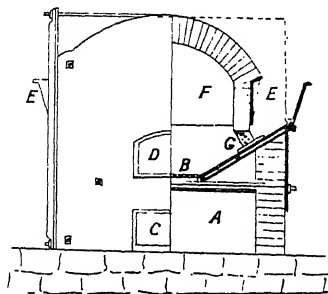


Fig. 303.

\* *Colo. S. Sch. of Mine Scient. Quarterly*, vol. ii., No. 1, p. 1; *E. and M. J.*, Oct. 7 and Dec. 9, 1893; May 23, June 20, and Aug. 22, 1896; and Hofman, *Metallurgy of Lead*, 1906, pp. 139-148.



cent., Pb less than 1 per cent. The fumes from the blowing-up furnaces are sent to a bag-house, together with those from the blast furnace next to be described.

The *blast furnace* used for smelting the clinker from the blowing-up furnace, together with a certain amount of coppery and other sulphide ores, is of peculiar construction, and is shown in Figs. 304 and 305. It measures 36 inches by 108 inches at tuyeres, but the height of charge columns in the centre is only about 12 inches, and even at the sides does not exceed 30 inches. In the Figs., A is the base or filled-up crucible of the furnace, B are the highly inclined water jackets, C is the wind chest, which is continuous along each side of the furnace, and from which the blast passes through the jackets by means of the twelve slot-shaped tuyeres, D, each 8 inches long by  $1\frac{1}{4}$  inches wide, being controlled by means of blocks attached to long rod-handles, E are the charge pockets which are kept always full of charge that slides down

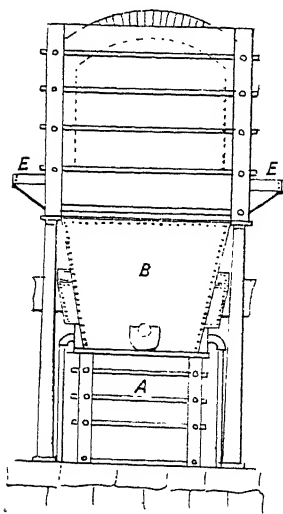


Fig. 304.

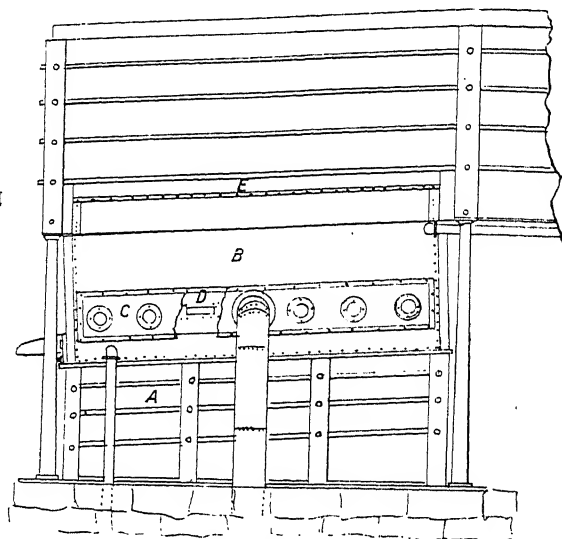


Fig. 305.

the jackets as the column inside the furnace is melted away, and keeps an angle of about  $45^\circ$ . The bottom of the furnace is inclined from one end to the other at about 1 inch to the foot, and the slag spout at the lowest point delivers the molten products into an overflow pot, and thence the slag into ordinary slag pots. The blast pressure is about 12 ozs. per square inch, the fuel consumption from 6 to 15 per cent., according to the amount of zinc to be volatilised, and the quantity of charge put through also varies from 40 to 75 tons per twenty-four hours, according as the percentage of zinc in it ranges from the maximum of 20 per cent. down to, say, 12 per cent. The charge mixture contains generally Zn from 17 to 20 per cent., Cu 2 to 4 per cent., Pb 3 to 10 per cent., S 15 to 20 per cent., CaO about 10 per cent., the remainder being silica and iron oxide, though occasionally, when the charge is unusually ferruginous, no lime is added. The slags vary in com-

position, the usual limits being  $\text{SiO}_2$  (insoluble) from 36 to 40 per cent.,  $\text{FeO}$  ( $\text{MnO}$ ) 31 to 47 per cent.,  $\text{ZnO}$  7 to 14 per cent.,  $\text{CaO}$   $2\frac{1}{2}$  to 18 per cent.,  $\text{Ag}$  0.25 to 1.5 ozs. per ton, but even more extreme limits are occasionally reached. The matte-fall is usually arranged to be about 8 to 10 per cent. of the ore charge; its percentages of copper, gold, and silver, of course, vary according to those of the original ore charge, but, when possible, the composition of the ores bought is arranged so as to produce a matte running about 30 to 40 per cent. copper, 2 to 4 per cent.  $\text{Zn}$ , 75 to 125 ozs. of silver, and 1 to 2 ozs. of gold per ton. The slags produced contain much zinc oxide uncombined with silica, and probably this exists as a ferrite, as suggested in Chapter vii.

Besides true fume, to save which is the principal object of the process, a large amount of flue-dust is made both in the sintering-hearth and in the blast furnace, and this is returned to the process, the true fume from the sintering hearth containing on an average no more than 5 ozs. per ton, and that from the blast furnace 8 ozs. per ton. Volatilisation of silver is increased by a high proportion of galena in the charge, by the presence of arsenic and antimony, and by a calcite gangue, and decreased by high proportions of pyrites and copper ores. The total loss of silver per ton of ore smelted varies from  $\frac{3}{4}$  oz. up to 3 ozs.; lead and gold usually show plus returns on the assay values.

The gases from sintering and smelting furnaces are drawn off by means of Sturtevant fans, after passing through a brick chamber to mix them and equalise their temperature, and forced through an iron chamber, where they drop some of their dust contents. They next pass through a pair of rectangular sheet-iron cooling flues, 8 feet  $\times$  3 feet  $\times$  1,400 feet long (like Fig. 203), in which most of the remaining dust is settled out, the amount averaging 3 per cent. of the weight of ore smelted, and 25 square feet of cooling surface being allowed per square foot of grate area of the sintering furnaces. From the cooling flues the gases pass to one of a pair of bag-houses, used alternately, the general construction of which is similar to that shown in Fig. 32 (Chap. iv.).

Each bag-house contains 1,500 bags, 21 feet long and 20 inches in diameter. The bags are made either of calico, costing  $3\frac{1}{2}$ d. per yard, which lasts from 18 to 24 months, or of woollen flannel, costing 1s. 8d. to 2s. 1d. per yard, which, however, lasts from six to ten years. The temperature of the gases must not exceed  $90^\circ \text{C}$ . for the cotton bags, but may reach  $120^\circ \text{C}$ . for the woollen ones. For each square foot of grate area in the sintering hearths 200 square feet of cotton filtering cloth are required, and if twice shaken in the twenty-four hours each square yard can be relied upon to collect 10 ozs. of fume per day, while the woollen cloth is more efficient, and will collect 1 lb. of fume per square yard per day. The composition of the raw fume is  $\text{PbS}$  12 per cent.,  $\text{PbSO}_4$  with some  $\text{PbSO}_3$  30 per cent.,  $\text{ZnO}$  14 per cent.,  $\text{ZnSO}_3$  40 per cent.,  $\text{SO}_2$  2 per cent.,  $\text{C}$  and  $\text{As}_2\text{O}_3$ , &c. (insoluble), 2 per cent.

In order to eliminate traces of arsenic and cadmium, antimony, mercury, and other elements, to burn off finely-divided carbon, and to convert all the lead into basic sulphate, and all the zinc into oxide, it is necessary to "refine" the fume by means of an oxidising roast. This is carried out at a temperature of  $815^\circ \text{C}$ . in cast-iron cylinders or retorts, 10 feet long and 12 inches in diameter, within each of which revolves a long cylindrical spiral

carrying upon its outer edge four longitudinal bars. The fume takes about twenty minutes to pass through the retort, and comes out much denser than it went in, four volumes of raw fume being compacted into one volume of pigment. Its composition is zinc oxide with lead sulphate and basic oxy-sulphate, and its average analysis Zn 47.33 per cent., Pb 24.92 per cent., S 2.96 per cent.,  $\text{Fe}_2\text{O}_3$ , &c., 0.45 per cent., O 24.31 per cent.

The cost of treatment is not made public, but is certainly under 25s., and probably under 21s. per ton at Canon City, with coke at 21s. and slack coal at 2s. per ton. The process would doubtless have met with much wider acceptance, but for the fact that the pigment produced, although bluish-white in colour and of fair covering power, does not find a ready sale; in fact, according to Bartlett,\* about 50 tons per day (produced from 200 tons of ore) is the maximum that the American market will absorb.

*Zinc-lead Pigment Manufacture at Coffeyville, Kas.*† Any mixed zinc-lead sulphides can be used for the process, but the minimum content of both metals together should be 30 per cent., and the relative proportion of the two metals is preferably about 2 zinc to 1 lead, giving a pigment containing about 2 parts of zinc oxide to 1 of basic lead sulphate.

The ore, after crushing to  $\frac{1}{4}$  inch, is roasted in McDougall furnaces from an original sulphur content of 15 to 25 per cent down to about 3 per cent., and allowed to cool thoroughly. It is then charged upon the 6 feet by 12 feet perforated grates of brick-built furnaces in two blocks, two rows of nine furnaces each, back to back, with a common flue, forming each block. The grates are of cast-iron in strips 8 inches wide  $\times$   $1\frac{1}{2}$  inches thick and 6 feet long, perforated with holes  $\frac{1}{4}$  inch in diameter and  $1\frac{1}{2}$  inches apart, through which the blast passes. In operating these furnaces a layer of coal is first charged evenly upon the grates, and when this is burning freely a layer of roasted ore intimately mixed with coal is thrown in and spread evenly. The blast first expels the reduced metals, and then re-oxidises them, and the whole of the fumes from the 18 furnaces pass through a brick combustion chamber, where any particles of carbon are burnt off. From this chamber another fan draws off the fumes through a sheet-steel cooling pipe, 700 feet long, and then forces them through the bag house, which consists of 18 horizontal conduit pipes, each with 50 nipples above for the same number of 30-foot bags suspended from the roof, and 17 hoppers below, from which depend the same number of bags, 8 feet long. Every four hours the long bags are shaken, and the filtered fume falls into the short collecting bags, which are removed every twenty-four hours, and the contents put on a conveyor, which takes it to the packing-room.

#### SMELTING PROCESSES.

A whole series of processes have been patented at different times by Angell, Ellershausen, Fry, and others, depending upon the smelting of the mixed ore in a blast furnace to yield lead in the metallic form, while zinc was kept in the slag by means of alkalis, partly as zincate and partly as sulphide, and recovered therefrom subsequently by a variety of means.

\* *Colo. S. Sch. of M. Scient. Quart.*, vol. ii., No. 1.

† *M. S. P.*, Oct. 31, 1908, p. 605.

4. The **Ellershausen** process,\* in one of its most recent forms, consisted in smelting the mixed sulphide ore raw in a cupola furnace with a high blast and hot top, so as to volatilise most of the lead and zinc and part of the silver, the remainder of the silver, together with the gold and copper contents of the original ore, being collected in lead bullion and matte, which were separated from the slag as usual. The fumes from the furnace were drawn off by a fan and churned up with water sprays, yielding (1) a solution of zinc sulphate, and (2) a sludge of  $\text{ZnS}$ ,  $\text{ZnO}$ ,  $\text{PbSO}_4$ , &c., which was filter-pressed and dried on floors. The broken-up cakes of sludge were then charged into a boiling solution of  $\text{NaHO}$  in an iron pot; the lead was reduced instantly to metal carrying with it the silver. The pot residues, after cooling, being broken up and passed to a leaching tank, yielded a precipitate of zinc sulphide and soda liquors which, upon being mixed with the solution of zinc sulphate obtained from the first washing of the fume precipitate zinc as hydrate, carbonate, and sulphide. The residual solution of sodium sulphate, after evaporation, is treated in the black-ash furnace for regenerating caustic soda.

5. The **Fry** process employed by the Zinc Corporation at Ellesmere Port consisted of the following operations:—

(1) *Roasting* the crushed ore down to about 5 per cent. sulphur, the furnaces employed being of the Godfrey pattern.

(2) *Briquetting* the roasted ore together with about 25 per cent. of its own weight of salt cake, and 5 per cent. of quicklime as binder.

(3) *Smelting* the briquettes in an ordinary lead blast furnace of the old low pattern, adding ironstone and slag as indicated by the composition of the charge, in order to produce a good ferruginous slag carrying all the zinc, separation of matte from slag being prevented by the alkali. The blast pressure employed was about 24 ozs., and lead was drawn off by an ordinary syphon, while the slag, containing much zinc as sulphide and probably as zincate of soda, was tapped from each end through overflow pots into ordinary slag pots.

(4) *Dezincing* the slag by breaking it into pieces of nut size, mixing with 10 to 15 per cent. of anthracite fines or coke breeze, and charging into open hearth regeneratively gas-fired furnaces with basic hearths of dolomite. Upon heating to a temperature of  $1,700^\circ$  to  $1,900^\circ$ , rabbling and poling with green wood, a portion of the zinc was reduced and volatilised, condensing in flues and settling chambers (and to a considerable extent in the chequer work of the regenerative appliances), as zinc oxide. Upon thus drawing off a portion of the zinc from the slag and lightening its specific gravity, a considerable separation took place of low grade alkaline iron matte, which contained practically all the lead and silver contents of the original slag, and, after crushing and roasting, was returned to the blast-furnace charge, while the now clean slag was thrown away. The zinc oxide recovered contained on an average 60 per cent. of zinc, but the quantity recovered was much smaller than had been hoped, and the choking up of the chequer work (erected in close proximity to the hearths) was serious; water-spray towers, moreover, had to be erected to complete its collection. The recovery of silver was good, but there was considerable volatilisation loss of lead. During the year 1897 10,872 tons of ore were treated by this process, the average

\* *E. and M. J.*, Aug. 11, 1900.

composition being Pb 26 per cent., Zn 26 per cent., and Ag 32 ozs. per ton, and the profit on ore of this grade purchased at a cheap rate was nearly £1 per ton; the recovery of lead and silver was said to be 90 per cent. each, and that of zinc 70 per cent. The process was abandoned, not so much upon its merits, as in consequence of extravagance and mal-administration in connection with the works.

**6. The De Laval Process.**—This process has been in use for some years near Trondjem, in Norway, and its introduction into other districts is said to be probable.

The process consists in forcing the finely pulverised ore mixture by means of jets of producer gas and air into a chamber of approximately cylindrical form, lined with magnesia brick, the jets being situated near the top of the chamber and arranged tangentially, so that the flame with its ore-charge circles round the furnace, whence the name "Cycloldal" furnace. An oxidising atmosphere is maintained, so that the zinc and lead contents of the ore are volatilised by the intense heat and converted into oxides, which issue suspended in the current of furnace gases from a flue near the bottom of the furnace, whence they are drawn off through a cooling dust-flue, followed by a dust chamber, and finally forced through bag filters by means of a fan. The silica, iron, and other gangue matters contained in the ore are fused, and fall to the bottom of the furnace, where a molten bath is formed which separates into two layers of slag and matte as in the crucible of a blast furnace, the matte containing practically all the copper, silver, and gold contents of the ore.

The fume caught in the bags, which consists chiefly of zinc oxide and lead sulphate, is charged into a similar "Cycloldal" furnace, in which a reducing flame is maintained, so that the oxides are reduced to the metallic condition; part of the lead forms a bath in the crucible of the furnace, the remainder accompanies the "zinc-grey" or "blue powder," which is drawn off by a fan, and forced through a bath of molten lead, where part of it is condensed, the rest passing on to another bag-house, where it is all filtered from the furnace gases. The product is compressed and melted down.

**7. The Snyder process** \* of electric smelting aims at volatilising the zinc after reduction to metal, while the lead reduced simultaneously is absorbed in a bath of that metal. An electric furnace is employed, the hearth of which contains a pool of molten slag; below this open the inner legs of two U-shaped lead wells, the outer legs of which extend outside the furnace; while the inner openings, a foot or more apart, are covered by the pool of molten product, which itself is covered by a layer of incandescent carbon. The lead wells form the two poles or electrodes. The ore, dead roasted and mixed with carbon, is first preheated in a chamber situated above the feed tube in the roof of the furnace, the heating being effected by the waste gases from the furnace itself, composed chiefly of CO. It is then fed in a steady stream upon the incandescent carbon, by which the metals are reduced, the zinc volatilising and being carried away to condensing chambers by the current of CO produced by the reduction, while the lead condenses in droplets, and is absorbed by the bath in the wells. The temperature requires to be kept between 1,000° and 1,100° C., in order to avoid as far as possible volatilisation of lead with the zinc; and the slag made, therefore, must have

\* *Mining Magazine*, Nov. 1909, p. 220.

a low formation point, a good composition being  $\text{SiO}_2$  40 per cent.,  $\text{FeO}$  30 per cent., and  $\text{CaO}$  30 per cent. The lime required is best added in the form of quicklime, in order to avoid unnecessary loss of heat in the  $\text{CO}_2$  which would be expelled if limestone were used. The process has not yet passed the experimental stage.

8. The **Greenway** process aimed at eliminating the lead and silver contents of Broken Hill middlings and similar lead-zinc ores by roasting with salt at a low heat. The lead and silver were to be converted into volatile chlorides which could be condensed by suitable means, while the half-roasted residues, after leaching out sodium sulphate, would contain from 30 to 50 per cent. of zinc, and be in a suitable condition for distillation. The method, however, seems never to have reached the commercial stage.

## B. SOLUTION AND PRECIPITATION PROCESSES.

1. The **Grimm Process**.<sup>\*</sup>—This oldest and simplest of all wet processes, first tried by Grimm at Przibram, is still in use at Goslar (Harz) for treating the Rammelsberg ores. The ore mixture contains  $27\frac{1}{2}$  to 30 per cent. blende, 9 to 12 per cent. galena, 1 to  $1\frac{3}{4}$  per cent. chalcocopyrite, 8 to 12 per cent. pyrites, 45 to 49 per cent. gangue, chiefly barytes and schist, and is roasted in heaps several times. After roasting, the fines (which alone are lixiviated, the lump ore being smelted direct) contain  $\text{ZnO}$  28 per cent.,  $\text{PbO}$  10 per cent.,  $\text{Fe}_2\text{O}_3$   $13\frac{1}{2}$  per cent.,  $\text{CuO}$  0.9 per cent.,  $\text{BaSO}_4$  24 per cent.,  $\text{CaO}$ ,  $\text{MgO}$ , and  $\text{Al}_2\text{O}_3$  5 per cent., and  $\text{SO}_3$  16 per cent. They are charged, 1 ton or more at a time, into revolving cylinders or drums 4 feet long and 3 feet diameter, made of boiler plate with cast-iron ends, and there leached three or four times. The first two washings are with still hot weak liquors from a previous charge, and when drawn off they have densities of  $50^\circ$  B. and  $20^\circ$  B. respectively; two further washings are then given with hot water.

The first two washings are heated to boiling and allowed to stand for a week in large lead-lined pans; after which time almost all the iron has crystallised out as a basic salt, together with much gypsum. The clear liquors are concentrated in boiling pans to  $50^\circ$  B., and either drawn off to crystallisers, in which the zinc sulphate separates out, or to tanks in which they are precipitated with barium sulphide, obtained by reducing the barytes, which is a common gangue in the district. A double decomposition results with formation of two insoluble precipitates—viz., zinc sulphide and barium sulphate. The mixture is run through filter presses, dried, calcined, and, after grinding, is sold as a white pigment of fair covering power under the name of "lithopone."

A modification of this process for preparing a mixed white pigment is that of Cowley,<sup>†</sup> according to which  $\frac{1}{2}$  to  $1\frac{1}{2}$  per cent. each of freshly prepared magnesia and finely pulverised salt are added to the mixed precipitate sludge before filtration. The dried precipitate is further mixed with 3 per cent. of ammonium chloride before ignition or calcination. The colour prepared in this way does not darken after long exposure to sunlight or to sulphuretted gases.

<sup>\*</sup> Schnabel, *Handbuch der Metallhüttenkunde*, vol. i., p. 327; also Kerl, *Min. Ind.*, vol. iv., 1895, p. 83.

<sup>†</sup> *Chem. News*, 1891, pp. 63 and 68.

"Lithopone" has good covering power, and does not continue to oxidise linseed oil as does white lead; it is, therefore, termed a non-siccative pigment. For outdoor work it seems unsuitable, because it is affected by exposure; but for indoor work it has some advantages, and, although absorbing light and becoming grey upon exposure to brilliant sunlight (a property of the zinc sulphide, which forms such an important part of it), the grey colour is lost again if shaded from direct sunlight, and it becomes brilliantly white in the dark. In theory the precipitate should contain about 30 per cent. ZnS and 70 per cent. BaSO<sub>4</sub>, but the commercial article sometimes contains about 25 per cent. ZnS, 7 per cent. ZnO, and 18 per cent. BaSO<sub>4</sub>, and at other times up to as much as 75 per cent. BaSO<sub>4</sub>. The presence of even a trace of iron spoils the colour, therefore the zinc solution must be carefully purified.

**2. The Parnell Process.**—This ingenious process was in operation at Llansamlet, near Swansea, from 1879 to 1883, but was commercially unsuccessful, owing, however, in part to causes quite independent of its merits. It comprised the following operations: (1) Roasting at a dull red heat; (2) leaching with water to dissolve ZnSO<sub>4</sub>; (3) heating in revolving lead-lined cast-iron pans with chamber sulphuric acid to dissolve ZnO; (4) leaching again with water to dissolve out ZnSO<sub>4</sub>; (5) mixing the sulphate solutions and passing over scrap iron to precipitate any copper present; (6) evaporating to a paste, and stirring in powdered blende in the proportion of one-fifth by weight of the ZnSO<sub>4</sub>; and (7) calcining in a muffle, whereby the following reaction took place—



The ZnO was sold to zinc smelters, the SO<sub>2</sub> from the muffle calciner, together with that from the ore roaster, was passed into lead chambers and converted into sulphuric acid. The leached residue after drying formed a readily saleable lead-silver concentrate sold to smelters. The losses of lead and silver are supposed to have been about 1 per cent., and that of zinc about 20 per cent. of the assay values.

The modification of this process, patented by Myalls and Wyatt,\* consists in dividing the mixed solution from the water and sulphuric acid leaches into three-fourths, which are evaporated to dryness, yielding zinc sulphate; and one-fourth, in which the zinc is precipitated by means of alkaline sulphide. The sulphate and sulphide are then mixed and heated in a muffle.

**3. The West Process.**†—Several hundred tons of mixed sulphide ores were treated by this process during 1889-90 at Thomasville, N.C.; but although the recovery of precious metals was practically complete, and the losses of lead and zinc only 20 per cent. and 30 per cent. respectively, the process was not a success under the conditions prevailing, which included an experimental and improved plant not well suited to the process. The following was the sequence of operations: (1) Roasting in a reverberatory; (2) cooling and wetting down, charging into false-bottomed tanks upon a layer of pebbles; (3) forcing the gases from the roasting furnace, together with a jet of steam, through the ore in the tank by means of a powerful blower for from six to twelve hours, by which means the zinc oxide present

\* *Min. Ind.*, vol. iii., p. 432.

† *Proc. Inst. Civ. Eng.*, vol. civ., p. 407.

was eventually changed into sulphate; (4) removing the ore to leaching vats and leaching with water; (5) precipitating the zinc from the neutral solution as hydrate by means of gaseous ammonia generated in stills by the action of caustic lime upon ammonium sulphate; and (6) regenerating the ammonia from the solution after precipitation of the zinc, by heating it with lime in the stills aforesaid.

The leached residue, containing all the precious metals and practically all the lead and iron of the original ore, was dried above the roasting furnace, and was then sold to lead smelters.

**4. The Storer Process.**—This ingenious process, which it was proposed to employ at the Illawarra Works of the Smelting Company of Australia, consists briefly in roasting, leaching out the zinc sulphate with dilute acid, peroxidising any iron present, filtering and then precipitating the zinc by means of calcined magnesite as a hydrate, which is run through a filter press and then calcined to oxide. The magnesium sulphate mother liquors are evaporated to dryness and calcined at a red heat, regenerating magnesia, which is used over again for precipitating another tankful of zinc sulphate solution. The sulphuric acid vapours given off from the calcining furnace are condensed and used for leaching out the roasted ore, thus making the process regenerative, both as to solvent and precipitant.

All the above processes, as well as those of Croscmire\* and Emmens,† depend upon sulphatisation of the zinc.

**5. The Schnabel Process.**‡—This involves roasting the ore at a cherry-red heat to convert the zinc into oxide, which is then dissolved out by a solution of ammonium carbonate; it has never been tried on a commercial scale, and even if the cost of the plant and of the necessary manipulations did not preclude its use upon a large scale, the inevitable leakages of ammonia would be likely to do so.

**6. The Spence process**, unlike the foregoing, aimed at the separation of the lead, leaving the blende unaltered, together with the precious metals. It consisted in acting upon the finely powdered raw ore with hot hydrochloric acid, which dissolves galena before it attacks blende. The lead chloride was dissolved out by hot water and the lead precipitated from it by calcium carbonate, while the lead-free blende (carrying still all the precious metal contents of the ore) was roasted and distilled for zinc in the ordinary way, the residues being returned to the silver smelters. This process was worked for some years by the late Mr. Spence, but proved unsuccessful commercially.

Some years ago the author experimented with this process on a dressed slime product containing 48 per cent. zinc and 14·7 per cent. lead. By once boiling up in covered wooden vats provided with agitator arms, with hot commercial HCl in the proportion of 100 gallons to 1 ton of dry ore, syphoning off and washing successively with several lots of hot water, amounting in all to about 12 tons of washings for each ton of solid material, the lead in the residue was reduced to under 2 per cent., and the zinc raised to 57½ per cent. Of the dissolved lead, two-thirds were crystallised out as chloride, and the other third precipitated as chromate. In the neighbourhood of chemical works where hydrochloric acid can be purchased cheaply this

\* *E. and M. J.*, Sept. 29, 1888; Feb. 9, 1889. † *Min. Ind.*, vol. iii., p. 431.

‡ *v.* Chapter xv., where its employment for the treatment of zinc crusts is described, also Schnabel, *Handb. der Metallhüttenkunde*, vol. i., p. 564.



process, carried out, however, not in vats, but in revolving barrels, is not only feasible for separating lead from zinc in a high-grade concentrate, but can be carried out at a profit.

Patents have been taken out by Siemens & Halske, Borchers, and several other inventors, for extracting zinc as chloride by means of chloride gas.

7. **Borchers** \* proposed to treat mixed zinc-lead ores in revolving lead barrels at temperatures of 30° to 40° C., together with solutions of NaCl or MgCl<sub>2</sub>, chlorine gas being passed through the trunnions. By leaching the residues with hot mother liquors, and subsequently with hot water, chlorides of Pb, Zn, and Ag are extracted, and free sulphur is distilled from the residue by Schaffner's method with superheated steam. The residue, which contains most of the blende and silver, is treated as an ordinary zinc ore, and the residues smelted for silver as usual. A Broken Hill ore with 14 per cent. Pb, 31 per cent. Zn, and 0.059 per cent. Ag yielded in solution as chlorides the whole of the lead, 17.5 per cent. of the Zn, and 0.011 Ag, leaving in the residue 13.5 per cent. Zn and 0.053 per cent. Ag. The final residue, after distillation of Ag, contained 39.1 per cent. Zn, and was 34.5 per cent. of the weight of the original ore.

The chloride solution was to be purified by addition of ZnO, or of a high-grade roasted blende, boiled down and electrolysed (see next section, the electrolytic processes of Swinburne, Maxwell-Lyte, and Borchers).

8. The **Hoepfner** chloride process will be found referred to in the section on electrolytic methods.

9. **Bisulphite Processes**.—Many years ago *Hoepfner* patented and used a process, the essential features of which were—(1) Roasting "dead" in muffle furnaces and collecting the SO<sub>2</sub>, condensing the last traces by means of scrubbing towers; (2) "Gassing" the roasted ore by means of the SO<sub>2</sub> gases produced in the roasting furnace; (3) Leaching out zinc bisulphite; (4) boiling the solution of zinc bisulphite [ZnH<sub>2</sub>(SO<sub>3</sub>)<sub>2</sub>] by which SO<sub>2</sub> is given off to be used for "gassing" another charge of ore, and insoluble zinc monosulphite precipitated, which was oxidised to sulphate by a method referred to on a later page. This method has been revived and thoroughly investigated by Sulman and Picard, who dispose of the monosulphite by simple dissociation at a temperature of 550° to 650° in closed cylinders, the products being impure zinc oxide for zinc smelters, saturated solution of sulphurous acid, and free SO<sub>2</sub> gas, the two latter being used for gassing new charges of roasted ore, while by adopting modern appliances the process has been in other ways brought up-to-date and made practical.

After a great deal of experimenting with *Hoepfner's* original method of decomposing the bisulphite solution by boiling off the SO<sub>2</sub>, and so precipitating the insoluble crystalline monosulphite, the inventors prefer to effect precipitation by incorporating with the solution finely divided zinc oxide produced by heating the monosulphite in a muffle at a low temperature. Zinc bisulphite being only soluble in solutions carrying 1½ per cent. of free SO<sub>2</sub>, this excess of free SO<sub>2</sub> has also to be neutralised by ZnO, but in practice it is found that 10 per cent. excess over the equivalent required by the bisulphite present is sufficient.

The *Sulman and Picard* process, as worked out by the inventors, therefore, consists of the following operations:—

\* *Min. Ind.*, 1902, vol. xi., p. 622.

(1) Roasting the ore, crushed to from 16 to 20-mesh size, in some form of mechanically rabbled furnace (M'Dougall, Wedge, or other type) muffled, at all events as regards all but the finishing hearth, the temperature employed not exceeding  $750^{\circ}$  to  $825^{\circ}$  C. Too high a temperature, besides causing volatilisation losses of lead and silver, increases the production of insoluble zinc ferrite; too low a temperature leaves too much zinc sulphate undecomposed. The formation of zinc sulphate and of zinc ferrite are the two principal difficulties of the process; from 5 to 10 per cent. at least of the zinc contents of the ore remain as sulphate, which, although soluble, demands separate treatment by double decomposition with calcium bisulphite, or precipitation with milk of lime, forming a mixed zinc-lime mud; while another 5 to 10 per cent. of the zinc contents take the form of zinc ferrite, which remains with the other insoluble matter contained in the ore during the leaching operation. The production of zinc ferrite can be to some extent controlled by keeping the roasting temperature low (under  $850^{\circ}$  C.), by having the bulk of the ore fairly coarse (between 25- and 80-mesh), and by "balling" the fines below 80-mesh with calcium sulphate produced at a later stage through precipitation of zinc sulphate with calcium bisulphite or milk of lime, before charging them into the roasting furnace. The roaster gases contain from 5 to 7 per cent.  $\text{SO}_2$ ; they are condensed in scrubber towers of any convenient form, the resulting liquors being used for leaching, while a portion of the gas may be absorbed by milk of lime, forming calcium bisulphite, which is used for transforming any zinc sulphate leached out from the ore into bisulphite by double decomposition.

(2) Leaching the roasted ore with  $\text{SO}_2$  solution from the scrubbers, passing through it at the same time strong  $\text{SO}_2$  gas from the decomposition of zinc monosulphite by heat; this leaching may be conducted either in barrels with gas pipes through the trunnions and special arrangements for gas-tight joints, in Brown agitation tanks closed at the top and furnished with pumps for collecting the unabsorbed gas, and again forcing it through the pulp, or still more simply in spray towers. The zinc is dissolved out as bisulphite (with some sulphate), except such portion as was converted by the roasting into zinc ferrite, which remains with the lead sulphate and insoluble constituents of the ore. In practice it is found convenient to treat a fresh ore charge with weak scrubber liquor, the  $\text{SO}_2$  contents of which are readily absorbed by it, until most of the zinc is converted into monosulphite, the weak liquors are then turned into a new ore-charge, and "gassing" is continued with the strong gases from the subsequent stages of the process until all the monosulphite is converted into soluble bisulphite. Towards the end of the leaching operation a small quantity of copper is extracted, and can be precipitated from the liquors by addition of zinc-dust or "blue powder"; if much copper be present, however, the bulk of it remains with the residue as oxide, and can be extracted by washing with dilute sulphuric acid, followed by precipitation as above. Some iron, manganese, and silica are extracted by the relatively strong sulphurous acid employed, so that the product of zinc oxide contains from 10 to 20 per cent. of impurities, equivalent to a tenor of 60 to 70 per cent. of zinc. Inasmuch as this product, however, is used for spelter-making, the presence of these impurities, so far from being a detriment, becomes an advantage, by increasing the density of the charge and retarding the speed of distillation, which would otherwise be abnormal.

(3) *Separating* the residue from the solution by means of settlement followed by *centrifugal drying* for the coarse, and subsequent re-settlement with *filter-pressing* for the slime which accompanies the solution from the centrifugals.

(4) *Agglomerating* the residue by "blowing up" with 10 per cent. of coal slack in a suitable grate-furnace with a light blast, producing a cinder, which contains most of the lead and practically all the silver in the original ore. A considerable proportion of the zinc ferrite produced in the roasting furnace is reduced, and the zinc contents are expelled as a fume of zinc oxide together with some lead. The fume is condensed in appropriate fume-chambers, and returned to the dissolving vats, where the zinc is extracted as bisulphite, while the lead returns to the insoluble residue as sulphate. The following analyses show the composition of ordinary lead-zinc ores from Broken Hill submitted to the process, and of the sintered residue for the blast furnace after extraction of the zinc.

	Original Ore.	Agglomerated Residue.
Lead, . . . . .	15·35	17·2
Zinc, . . . . .	14·65	1·6
Insoluble, . . . . .	45·31	63·1
Sulphur, . . . . .	11·26	1·4
Iron, . . . . .	6·10	..
FeO, and Al <sub>2</sub> O <sub>3</sub> , . . . . .	..	9·0
MnO, . . . . .	5·51	1·4
CaO, . . . . .	..	2·5
Unconsumed carbon, &c., . . . . .	..	3·5
	<hr/> 98·18	<hr/> 99·7
Silver, ozs. per ton, . . . . .	10·12	12·7

On a large scale this sintered product usually contains about 4 per cent. zinc, which is not enough to interfere with its blast-furnace treatment.

(5) *Precipitating* the zinc monosulphite by incorporating with the bisulphite solution finely-divided zinc oxide from the decomposition of a previous charge of monosulphite. This incorporation was formerly effected in a tube mill, but it is now found quite sufficient to agitate the zinc oxide with the solution by passing it through a centrifugal pump, which so churns up the lumps of zinc oxide as to effect their rapid solution and reprecipitation as monosulphite. As already stated, the quantity of ZnO required is only 10 per cent. in excess of the molecular equivalent, so that somewhat more than one-half of the zinc oxide produced in the calcining furnaces returns to the next cycle.

(6) *Drying* the monosulphite precipitate in a centrifugal dryer, the cakes from which contain, in addition to adhering moisture, some 23 per cent. of water of crystallisation (2ZnSO<sub>4</sub>, 5H<sub>2</sub>O). In order to avoid oxidation to sulphate, which would be reduced to sulphide in the subsequent distillation of the oxide, and so occasion a loss of zinc, the dried monosulphite should be transferred to the retorts as quickly as possible.

(7) *Calcining* the zinc monosulphide in iron retorts at a temperature above 550°, but not exceeding 650° C., the time required being about four hours. The water of crystallisation is driven off in several stages, the last molecule not being given off till a temperature of 300° C. is reached. The products are a strong 10 per cent. solution of SO<sub>2</sub>, which is used for leaching

ore-charges, but which contains only one-fifth or less of the gas expelled, and  $\text{SO}_2$  gas of such strength and purity that it may be sent to a "contact" plant for the production of sulphuric acid. The bulk of the  $\text{SO}_2$  produced from the ore in roasting being returned in leaching, as is also that produced by the decomposition of the zinc bisulphite, the various losses of sulphur in the process, including those in lead and calcium sulphates sent to the sintering furnace, are covered by only a portion of the  $\text{SO}_2$  obtained from the dissociation of the monosulphite; the balance of this is, therefore, available for utilisation in any convenient way, whether as sulphuric acid or as calcium bisulphite.

(8) *Reduction* of the zinc oxide in distillation furnaces. This presents no difficulty, the calcined zinc oxide with its 15 to 20 per cent. of impurity is much more dense than zinc oxide produced as fume, and behaves exactly like a high-grade roasted blende. In practice the zinc oxide produced in the process would ordinarily be sold to zinc smelters rather than treated on the spot.

It may be remarked that the percentage of insoluble in the agglomerated residue being too high for economical treatment by smelting, it would ordinarily be advisable to effect a partial concentration of the original ore, so as to eliminate part of the gangue, and reduce the bulk of material submitted to what must necessarily be a somewhat expensive process; this, however, would not apply to such ores as those of Rosebery and Mt. Reid, which are low in silica. The inventors estimate the total costs of treatment up to the production of zinc oxide and agglomerated lead-bearing residue at from 18s. 4d. to 30s. per ton, but on a commercial scale it seems probable that these figures might be exceeded.

More recently the original Sulman-Picard bisulphite process has been modified by other investigators, and in order to simplify it the cyclical return of  $\text{ZnO}$  has been given up, as well as the recovery of  $\text{SO}_2$  from the decomposition of zinc monosulphite, which formed such an important feature of the original process.

As now presented (1910) the process embodies the following operations:—

(1) Roasting and leaching of the roasted ore in spray towers with the roaster gases, which it is claimed are sufficient to dissolve out the whole of the zinc as bisulphite without any extraneous  $\text{SO}_2$ .

(2) Precipitation of the dissolved zinc bisulphite with milk of lime, giving a zinc-lime mud, composed of zinc monosulphite and hydrate with calcium sulphite and sulphate, the zinc contents of the product in a dry state ranging from 20 to 25 per cent. zinc.

(3) Briquetting the zinc-lime mud with coal in the proportion of 2 lbs. to every 1 lb. of zinc present, and "blowing up" on a sintering grate, by which means the bulk of the zinc is oxidised and condensed, first dry in ordinary brick chambers and then wet in spray chambers with the help of Körtling water sprays. Condensation is practically complete, and the product of zinc oxide with some monosulphite is said to contain over 60 per cent. Zn with less than 2 per cent.  $\text{CaO}$  and very little lead, most of which remains in the residue from the spray towers as sulphate.

(4) Distillation of the zinc oxide for spelter in the usual way.

(5) Smelting the residue from the extractor towers, which contains most of the lead and copper, and nearly all the precious metals in the original ore, by ordinary methods.

Although, however, the process has been apparently simplified by these modifications, it by no means follows that it has been really improved from a commercial point of view, since in some cases the value of the  $\text{SO}_2$  recovered by the original process would more than cover the extra cost introduced by cyclical rehandling of one-half of the  $\text{ZnO}$ . Moreover, in some localities the cost of the extraneous milk of lime would be an important consideration, whilst an additional source of loss is introduced through the necessity of submitting the whole of the zinc product to the blowing up operation, since some zinc is always retained in the calcareous sinter produced by this operation.

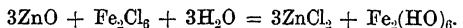
### C. ELECTROLYTIC METHODS.

Most of the processes which employ electrolysis for isolating the zinc separated in solution from a mixed ore are carried out upon a solution of the electrolyte (either zinc chloride or sulphate), but, as will be seen, there are others (Maxwell-Lyte, Swinburne, and Borchers) which employ fused zinc chloride as electrolyte. The methods with solutions of zinc sulphate or chloride will be first described.

1. The **Létrange** process consists in roasting the ore and leaching it with sulphurous acid, obtained by passing the furnace gases through water. The solution, containing zinc sulphite and sulphate, passes to tanks in which it is electrolysed, regenerating free sulphuric acid, which is applied to the leaching of another charge of ore until accumulated lime, iron, and other impurities render it necessary to start again with clean water. Practical difficulties in the electrolysis of acid zinc sulphate solutions prevented the process from attaining even technical success.

2. In the **Siemens & Halske** process the ore is roasted and leached with dilute sulphuric acid to obtain a sulphate solution for electrolysis. The liberation of free sulphuric acid, which proved fatal to the Létrange process, is overcome by Messrs. Siemens & Halske by the addition of ferrous sulphate to the electrolyte. This substance takes up oxygen as fast as it is liberated by the electric current at the anodes, producing ferric oxide which again combines with the sulphuric acid set free to form ferric sulphate, the combining power of iron for sulphuric acid being 50 per cent. greater in the ferric than in the ferrous condition. The solution of ferric sulphate then passes forward to the leaching tanks, where it dissolves more zinc, acting not only upon the oxide of that metal, but also to some extent upon unaltered sulphide; iron in the solution being partly precipitated in the interstices of the leaching charge and partly reduced again to ferrous sulphate, for which purpose, however, a certain amount of iron pyrites in the ore is found to be very desirable, if not, indeed, absolutely necessary.

3. **The Ashcroft Process.\***—The difficulties met with in producing a coherent deposit by electrolysis of a zinc sulphate solution led this inventor to adopt in his original process, both at Broken Hill and at Grays, Essex, a chloride solution produced by leaching the roasted ore with a solution of ferric chloride containing 10 grains of iron per litre, when the following reaction took place:—




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\* *Private Notes*, 1896; also Threlfall, *Australian Mining Standard*, Jan. 20, 1897, and Ashcroft, *Trans. Inst. of Mining and Metallurgy*, vol. vi., pp. 282-337.

Besides the zinc, however, some iron, manganese, and silver always went into solution as well as lead, unless sufficient sodium sulphate were added to convert it into insoluble sulphate; while part of the iron was reduced only to the condition of  $\text{FeCl}_2$ .

In the solution from the leaching vats any excess of  $\text{Fe}_2\text{Cl}_6$  was first precipitated by addition of a little  $\text{ZnO}$ , and the solution was then pumped through filter presses to separate the residual siliceous matter of the ore, together with lead sulphate and the precipitated ferric hydrate. The cakes from the filter presses were broken up and briquetted, for smelting with lead concentrates and other ores.

It was found that a hard and firm deposit of zinc from a chloride solution could only be obtained when the electrolyte was circulated *rapidly*, and when instead of being acid it was slightly *basic*, this latter condition being insured by the addition of a little zinc oxide in the storage vats.

In working on a large scale, however, the ferric chloride liquor was found to be quite unmanageable, corroding rapidly any form of metallic pump, and owing to this and to other causes a reversion was made at the *Cockle Creek Works* to the use of sulphate solutions, which could be readily handled with centrifugal pumps made of hard antimonial lead. With sulphate solutions the carbon anodes had to be replaced by lead plates, which last twelve months, being then converted largely into peroxide. With this change it was finally found practicable to dispense with the iron anodes and their accompanying consumption of that metal, and to use lead anodes exclusively, by which means an acid solution was regenerated for leaching, containing free  $\text{H}_2\text{SO}_4$  instead of  $\text{Fe}_2\text{Cl}_6$ , the diaphragms in the cells preventing this free acid from interfering with regular deposition on the cathodes.

The principal difficulties met with in the operation of this modified process on B.H. ores at the Cockle Creek Works \* were the following :—

(1) Difficulty of roasting even a finely pulverised material so as to produce any sulphate of zinc whatever, all the sulphuric acid produced being generally insufficient to convert the whole of the lead present into sulphate; and the consequent necessity of providing sulphuric acid from other sources for leaching.

(2) Impossibility of leaching thoroughly by percolation, which rendered it necessary to adopt the costly processes of agitation and filtering in presses. Even in filter presses, however, unless the solutions are at a temperature of  $80^\circ \text{C}$ . or over, and unless the free acid contained is kept below 100 grains per litre, it is very difficult to filter the ore-mud owing to separation of gelatinous silica.

(3) Gradual accumulation of manganese in the solution, which not only interfered with the solution and deposition of zinc, but used up a large amount of sulphuric acid, since in order to keep the accumulation of manganese within bounds a portion of the solution had to be periodically run to waste, or treated by some other process for recovery of the zinc contents.

(4) Difficulty of maintaining the electrolyte in such condition as to produce a uniformly hard and coherent deposit. It seems to be proved that not only the presence of traces of lead, antimony, arsenic, and other metals, but also traces of organic matter from new vats, oil from the re-use of condensed exhaust steam in the boilers supplying live steam for boiling up the

\* Ashcroft, *Trans. Inst. of Mining and Metallurgy*, vol. vi., p. 282, *et seq.*

leaching vats, and other, as yet unexplained, conditions will instantly change the character of the deposit from a hard, coherent, mass, like electrolytic copper, to mossy arborescent growths, or even to a black spongy mud which is practically valueless.

(5) Difficulty of smelting the filter-press cakes, containing the lead and siliceous matter of the original ore together with 8 per cent. of residual zinc, without previous briquetting and drying, and the necessity of adding a very large amount of flux, now that, owing to the abandonment of iron anodes, there is no longer any addition from outside to compensate for the iron and manganese dissolved out from the ore in leaching.

Owing to these difficulties, to the cost of the various manipulations required, and to the fact that for a capacity of only 1,000 tons of ore per week no less than £250,000 must be expended on plant, the process has been abandoned, and it is highly improbable that it will ever come into use again.

4. **Mohr's** \* process consists in roasting the blende and treating with a solution of sodium bisulphate, thus obtaining a solution of zinc sulphate saturated with neutral  $\text{Na}_2\text{SO}_4$ , which is electrolysed, depositing metallic zinc at the cathode, while sodium bisulphate is regenerated at the anode. The solutions are said to be free from iron, and, owing to the presence of  $\text{Na}_2\text{SO}_4$ , are said to be readily electrolysed, while no mossy zinc is formed, owing to the fact that the free  $\text{SO}_3$  immediately forms sodium bisulphate. The process is said to be in use at some works in Belgium.

5. The **Cowper-Coles** process † consists of roasting as usual and leaching the roasted ore with water and dilute sulphuric acid so as to obtain a solution with 1 oz. of free sulphuric acid and 20 to 30 ozs. of  $\text{ZnSO}_4$  per gallon, which is freed from copper by zinc and electrolysed in vats with disc cathodes of aluminium and anodes of lead plates, the former of which are rotated slowly during the electrolysis. The current density is from 15 to 20 amperes per square feet, and the voltage at the cells is from 1 to 2. The residue in the leaching vats is leached with a solution of caustic soda (30 ozs. per gallon) to extract the lead as sodium plumbate, which may either be electrolysed or used for the manufacture of lead white, the former method being usually adopted.

Broken Hill ores from the British Mine, containing Pb 30 per cent., Zn 24 per cent., Fe 4 per cent., S 19 per cent., and Ag 0·06 per cent., were treated by this process at Hayle on a fairly large scale, the residue after leaching containing as little as  $4\frac{1}{2}$  per cent. Zn. The zinc was deposited in the mossy form, and was removed from the cathodes by wooden scrapers, the product containing Zn 54·88 per cent., O 12·55 per cent.,  $\text{SO}_3$  10·05 per cent.,  $\text{SiO}_2$  1·05 per cent., CuO 0·80 per cent., PbO 0·07 per cent.,  $\text{Al}_2\text{O}_3$  1·20 per cent.,  $\text{Fe}_2\text{O}_3$  0·60 per cent., CaO 0·45 per cent., and water and organic matter 13·35 per cent. The lead deposited from its plumbate solution on the revolving discs contained Cu 0·08 per cent., Sb and Sn 0·0004 per cent., Fe 0·0117 per cent., Zn 0·015 per cent., and Pb by difference 99·8029 per cent. From the residue after lead extraction silver is recovered by leaching with KCy and precipitation on silver sheets. In this process no attempt was made to get a coherent deposit of zinc, although it is well known that the world's market for zinc dust or mossy zinc is extremely limited, and the re-treatment of the mossy product involves heavy losses.

\* *Min. Ind.*, vol. vii., 1899, p. 749.

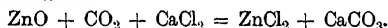
† *Mineral Industry*, vol. vii., 1899, p. 747.

6. The **Hoepfner** process for electrolytic deposition of zinc utilises a chloride solution prepared from blende or mixed zinc-lead ores in one of various ways.

One of the methods employed is as follows:—The blende is roasted in muffle furnaces with from 20 to 25 per cent. by weight of  $\text{CaCl}_2$  for twenty to twenty-four hours at a temperature not exceeding  $650^\circ$ , the coal consumption being 20 per cent.

The charge, as drawn from the furnace, and while still hot, is leached in iron vats with dilute waste liquors from a previous operation, yielding a solution containing about 10 per cent. Zn as  $\text{ZnCl}_2$ , besides the excess of  $\text{CaCl}_2$ ,  $\text{PbCl}_2$ ,  $\text{CuCl}_2$ , and other base metal chlorides. The lead is precipitated from the solution by means of zinc dust or mossy zinc; Fe and Mn are eliminated by addition of bleaching powder and marble dust, and the residual solution is electrolysed. In cases when the proportion of Pb is high much of the  $\text{PbCl}_2$  remains with the residue long after the extraction of  $\text{ZnCl}_2$  is complete. In such cases leaching is continued with hot water until no more can be extracted, and treated with scrap zinc to recover the lead. A certain proportion of the lead, however, is always left in the residue as insoluble basic sulphate, and, if there is any overheating in the roasting process, even silicate may be formed.

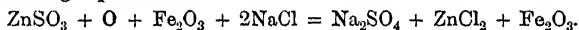
Another method for obtaining a chloride solution from roasted blende (said to be that in use at the **Winnington Works**) is that of agitating the roasted blende with hot calcium chloride solution, which is a bye-product of the ammonia-soda process, and passing  $\text{CO}_2$  under slight pressure, when the following reaction takes place—



Sometimes the roasted blende is agitated with the  $\text{CaCl}_2$  solution, forming calcium hydrate, and then  $\text{CO}_2$  is passed in a separate vessel; in this case the  $\text{CaCO}_3$ , which comes down in a fairly pure form, is known as “pearl hardening,” and is saleable to paper-makers.

Still another method of obtaining a zinc chloride solution is as follows:—The roasted ore is “gassed” with  $\text{SO}_2$  produced in the roasting, giving zinc bisulphite,  $\text{H}_2\text{ZnS}_2\text{O}_6$ , which is leached out. The solution is boiled to eliminate the free  $\text{SO}_2$  (which is used upon another charge), when the insoluble zinc monosulphite,  $\text{ZnSO}_3$ , is precipitated.

The precipitate is mixed with  $\text{NaCl}$  and  $\text{Fe}_2\text{O}_3$  (as oxygen carrier) and subjected to a current of warm air, when the sulphite becomes oxidised to sulphate, and by double decomposition forms the chloride, in accordance with the following equation:—



The  $\text{ZnCl}_2$  is leached out with the minimum possible quantity of water or brine solution, so as to leave most of the zinc sulphate undissolved and electrolysed.

This method seems more complicated than the two previously mentioned, but it is said to leave less zinc undissolved in the residue.

At *Furfort am Lahn*\* (Germany) zinciferous pyritic cinder from pyrites kilns was ground to 1.5 mm. size, mixed with 18 to 20 per cent.  $\text{NaCl}$ , and roasted in a muffle for twenty to twenty-two hours at a temperature not

\* *E. and M. J.*, May 16, 1903, p. 750.



exceeding  $650^{\circ}$ , the charge of cinder being 1,500 kgs., and the coal consumption 20 per cent. by weight. The still hot charge was leached in iron vats with mother liquors, yielding a solution containing 10 per cent. Zn as  $\text{ZnCl}_2$ , from which  $\text{Na}_2\text{SO}_4 \cdot 10 \text{H}_2\text{O}$  was deposited by freezing to  $-5^{\circ} \text{C}$ ., after which Fe and Mn were precipitated by addition of marble dust, and Pb, Cu, &c., by zinc dust. The residual solution with 20 per cent.  $\text{ZnCl}_2$  and 22 per cent. NaCl was electrolysed with discoidal rotating cathodes only partially immersed.

The vats were V-shaped, divided into seven anode cells and eight cathode compartments by means of a diaphragm of nitrated cloth; the anodes were of hard carbon, the cathode discs of zinc revolving on a central shaft. Each anode cell was closed at the top, and was provided with an outlet pipe of glass or hard rubber connected with the lead gas mains by means of an india-rubber tube to carry off the chlorine gas for making bleaching powder; at the bottom, of course, the anode cells were open to the cathode compartments, in order to permit free circulation. The electrolyte contained, as it entered each vat,  $9\frac{1}{2}$  to 10 per cent. Zn, and left with 2 per cent. The acidity was maintained at from 0.08 to 0.12 per cent. free HCl by addition of acid to the extent of 0.03 to 0.05 gramme HCl per ampere hour. The vats were connected in series, the current density employed was 100 amperes per square metre, and the terminal voltage of each vat was 3.3 to 3.6. Zinc deposited in dense form, and the current efficiency was 97 per cent. of the theoretical.

After thirty to thirty-five days the seven cathodes would have from 800 to 1,000 kgs. of zinc deposited on them, and were replaced by a new set and melted down in a reverberatory furnace with a coal consumption of 7 to 8 per cent., losing 1.5 to 2 per cent. in the melting process, and yielding a product with 99.97 to 99.98 per cent. Zn, 0.01 to 0.02 per cent. Pb, and traces only of Fe and Ti.

The *Mond* modification of the Hoepfner process carried out at the *Winnington Works* is said to employ corrugated rotating drums pressed together by means of springs, by which the surface of the deposited zinc is kept smooth, and the deposit is maintained at the required degree of denseness. The yield per H.P.-day of electric power employed is said to be 5 kgs. zinc and 15 kgs. bleaching powder, and the cost of production is said to be more than covered by the value of the zinc produced.

7. The *Maxwell-Lytle*\* process comprises leaching of the roasted ore with dilute sulphuric acid, and conversion of the sulphate into chloride by treatment with NaCl. The resulting zinc chloride is fused and electrolysed as in *Swinburne's Phoenix process* (*q.v.*), using a cathode of fused metallic zinc and carbon anodes.

8. The *Phoenix Process*.† (*Swinburne and Ashcroft*).— This process, which is completely cyclic, and aims at recovery of all valuable elements, comprises four stages:—

- (1) Displacement of sulphur by means of dry chlorine gas under pressure.
- (2) Precipitation of lead, copper, and silver from the fused chlorides by means of zinc and of iron and manganese by peroxidisation.
- (3) Separation of the fused chloride of zinc from the gangue, &c., by

\* *Min. Industry*, vol. vii., 1899, p. 748.

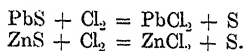
† *Trans. Inst. Min. Met.*, vol. ix., 1901, p. 378.

pouring off the clear supernatant portion and boiling up with water that which is mixed with gangue, the solution of  $\text{ZnCl}_2$  obtained being evaporated down and fused.

(4) Electrolysis of the fused zinc chloride.

Like many other cyclic processes, which on paper look pretty and promise complete recoveries, this ingenious process has in practice developed serious difficulties, and does not appear to have become commercially successful.

The *1st Stage* (displacement of sulphur by chlorine) is carried out in iron converters lined with firebrick of either the tipping or the fixed form with a taphole. The temperature at which the reactions become violent is  $600^\circ \text{C}$ ., below which there is loss of  $\text{Cl}$  by formation of  $\text{S}_2\text{Cl}_2$ . Above that temperature  $\text{S}_2\text{Cl}_2$  is completely dissociated, and the whole of the sulphur distils off free (except any which is burnt to  $\text{SO}_2$  by air accidentally entering the apparatus). The reactions are—



The ore, crushed only to moderate fineness, is fed in continuously through a trapped earthenware tube which dips below the surface of the bath of fused chlorides, and the chlorine gas (which is kept in iron tanks under 50 lbs. pressure) is blown in as required; the reaction being strongly exothermic, the required temperature is easily kept up. A certain amount of metallic chloride is volatilised and collected in the condensing flue together with the sulphur, from which it can be readily separated by washing with hot water, leaving the sulphur nearly pure.

The *2nd Stage* (substitution of zinc for lead) is carried out either in the converter itself, or preferably in a top-heated muffle furnace, provided with a charging tube for the zinc and a taphole for the reduced lead (and silver), the temperature necessary being only  $450^\circ \text{C}$ . After separation of the lead, iron and manganese are peroxidised and precipitated as oxides ( $\text{Fe}_2\text{O}_3$  and  $\text{MnO}_2$ ), which are removed with the gangue.

The *3rd Stage* (separation of gangue) is carried out when the mixed ore contains but little insoluble matter or iron by allowing the molten mixture to stand, and tapping or skimming off the clear chloride which remains above the stratum of suspended matter. When the proportion of gangue and iron is considerable no clear chloride can be removed, and the whole mass has to be submitted to the next operation, which consists of leaching with hot water, boiling down in lead-lined vessels under vacuum, and finishing in enamelled iron pans or top-heated tank furnaces. It is necessary to evaporate in a partial vacuum owing to the tendency to lose  $\text{HCl}$  and form basic chloride, and in order to remove the last traces of moisture, which if allowed to remain destroy the electrolysing apparatus.

The *4th Stage* (electrolysis of fused zinc chloride) is usually carried out in two stages. Fused  $\text{ZnCl}_2$ , if perfectly free from moisture, does not begin to fume at temperatures below  $400^\circ$ , and does not attack iron vessels or tools; but it affects iron if the slightest trace of moisture be present. In order to protect the main electrolysis vats, therefore, as well as the carbons (which also are destroyed by any moisture present), and to eliminate any traces of  $\text{Pb}$  and  $\text{Fe}$  still remaining in the  $\text{ZnCl}_2$ , and so produce a pure metal, the fused chloride is first partially electrolysed or purified in a special form of

open vat with a cheap form of carbon anode; and then, when chlorine commences to be evolved copiously, it is run into the permanent electrolysis vats or furnaces.

These consist of an outer steel tank, 6 feet in diameter, lined with 18 inches of ordinary Stourbridge firebrick. The cathode is a ton of molten zinc, which is always left in the bottom of the furnace below the taphole, connection with it being effected by means of a steel block buried in the brickwork, to which the current is taken by means of copper tubes. The anode is a cast-iron plate, with a group of 120 rods of hard carbon  $2\frac{1}{4}$  inches in diameter depending from it, and immersed 6 inches in the electrolyte. The current used is from 3,000 to 4,000 amperes at a pressure of 4 to  $4\frac{1}{2}$  volts, and each furnace turns out nearly a ton of zinc per week, nearly the full equivalent of metal corresponding to the current employed being deposited. The metal produced contains Pb 0.15 per cent., Fe 0.02 per cent., Zn (by difference) 99.83 per cent. The working temperature is  $425^{\circ}$  to  $525^{\circ}$  C. The Cl is drawn off from the furnace by means of an ebonite-coated fan, and is compressed by means of a pump, and liquefied at a low temperature caused by the re-evaporation of the gas itself, under a pressure of 50 lbs. per square inch.

In spite of the ingenuity of this process, and its pretty appearance on paper, after the lapse of something like nine years since it was first brought out, no commercial success would appear to have been attained.

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## CHAPTER XXI.

### FLOTATION PROCESSES.

A WHOLE series of processes for the separation of sulphide minerals from their stony gangue, and to a far less definite extent of one sulphide mineral from another, has come into use recently, all based primarily upon the fact that particles of sulphide minerals are less readily wetted by water than particles of gangue; while, on the contrary, the surface of the sulphide particles appears to have a positive attraction for gases and for grease in any form. It is a matter of every-day observation that particles of clean sulphide minerals when dry are readily floated by water, and the various flotation processes all depend upon different ways of accentuating the disinclination of the sulphide particles to allow their surface to become wetted by water.

The scientific explanation is somewhat obscure; whether the apparent repulsion between the sulphide surface and that of the water is a property essentially of the sulphide mineral or whether it depends upon the "surface tension" of the water is doubtful. Surface tension of a liquid is that property which enables it to acquire and retain a globular form when in contact with a surface which is not wetted by it. The free surface acts like a skin which exerts a certain resistance to severance by the introduction of foreign bodies, and this resistance is specific, varying with the body introduced, so that whereas gangue minerals readily break through the tension and become wetted, sulphide minerals are repelled, and the repulsion is increased when their surface is at all greasy or when coated with certain gases.

The oldest of the flotation processes is the original *Elmore* process, which depended upon oil alone for floating the sulphide minerals. Next came the so-called "acid" processes of *Potter* and *Delprat*, which use no oil, but depend upon the formation of bubbles of  $\text{CO}_2$  through the action of the acid upon carbonates in the ore, and the adherence of these bubbles to the particles of sulphide minerals in preference to those of the gangue, thus buoying up the former. The *De Bavay* process "gasifies" the dry ore by passing over it a current of furnace gases rich in  $\text{CO}_2$ , and then feeds it in a thin film on to an acid solution contained in *spitzkasten*, similar to those used for the *Potter* and *Delprat* processes.

The *Cattermole* or *granulation* process, tried at the Central Mine, Broken Hill, consisted in agitating the ore pulp with certain small proportions of oil and acid at a temperature of  $38^\circ$  and feeding into *spitzkasten*, when the concentrates overflow and the gangue is withdrawn below in the usual way. The concentrates are boiled with dilute caustic soda, which saponifies the oil, and this soap solution being added to a fresh lot of pulp, together with a fresh charge of acid, the oil, or rather fatty acid, is regenerated for re-use.

The new *Elmore vacuum* process utilises the air dissolved in the water,

## THE METALLURGY OF LEAD.

by the action of a vacuum, forms into bubbles; these attach themselves to the sulphide particles, and float them to the surface of the vacuum chamber. Small quantities of acid and of oil are employed intimately mixed with the pulp, in order to increase the attraction of the air bubbles to the particles of mineral by forming minute bubbles of  $\text{CO}_2$ , with which larger bubbles of air coalesce.

The newest of all the flotation processes is the *Macquisten* \* which uses neither oil nor acid, gas nor vacuum, but causes the sulphide minerals to float upon water simply by presenting the particles to the surface at a suitable angle. Each time that a group of particles of sulphide mineral are slid upon the surface of water at a flat angle a certain proportion will float, and the remainder, if slid upon the surface a sufficient number of times, will nearly all float eventually.

Huntington,† in the course of some investigations into the mechanics of flotation processes, experimented on the angle made between a smooth polished face of a mineral and the surface of a liquid, into which it is partially immersed. This angle is constant for a given substance and a given liquid, and is a measure of the degree to which the substance is wetted. The "angles of wetting" observed were for galena  $48^\circ$ , blende  $50^\circ$  to  $69^\circ$ , stibnite  $47^\circ$ , cerussite and quartz  $0^\circ$ —that is to say, these latter substances were completely wetted. If, therefore, particles of these different substances are immersed in water and at the same time brought into contact with bubbles of any gas, the extent of the surface of the latter, which can come into actual contact with the mineral particle, will be in direct proportion to the angle formed between the mineral and the liquid. The larger the angle the greater the area over which contact extends, and the greater tendency, therefore, for the bubble to lift the particle; whereas when the mineral particle is of such a kind as to be completely wetted even on a free surface, there is no tendency at all for the gas to attach itself to the particle.

In regard to the acid flotation processes, Huntington found ‡ that the gas evolved by calcite, and by other carbonates which are readily decomposable in acid, appears to have little or no tendency to attach itself to particles of metallic sulphides, and it is only the gas evolved by such carbonates, as blende and rhodocroisite, which require a heat of  $65^\circ$  for their decomposition, which has any flotative action upon the sulphides. This is affirmed by Ingalls. §

De Bavay has suggested || that  $\text{SiO}_2$  and S liberated, the former through the action of  $\text{H}_2\text{SO}_4$  on fluorides in presence of silica, and the latter through oxidation and decomposition of  $\text{H}_2\text{S}$  resulting from the action of the acid blende, play an important part in the acid flotation processes, by forming colloidal mediums or envelopes for holding the bubbles of evolved gases and attach themselves to the sulphide particles; but this explanation is somewhat far-fetched and unnecessary, the "angle of contact" between the sulphide particles and the wetting solution appearing to be sufficient to account for the attachment of the gas bubbles to these rather

*E. and M. J.*, Oct. 26, 1907, p. 765.

Paper read before the Faraday Society, Dec. 12, 1905; abstract, *E. and M. J.*, 7, 1906, p. 314.

*Loc. cit.* § *E. and M. J.*, Dec. 15, 1906, p. 1115.

D. K. Clark, *E. and M. J.*, Nov. 24, 1906, p. 906.

than to the particles of gangue. Absence of vibration is essential to all the flotation processes, for a slight shock is sufficient to detach the gas bubbles, and the sulphide particles then fall.

**The Potter Process.**—This process, in use at the Block 14\* and other Broken Hill properties, depends for flotation of the sulphide particles solely upon the generation of bubbles of carbonic acid gas from particles of calcite and chalybite contained in the ore; which as soon as disengaged, owing to the higher "angle of wetting" of the particles of metallic sulphides, leave the carbonates and attach themselves to these, floating them to the surface of the bath.

The dry tailings from wet concentration crushed to 30-mesh are fed into a square pyramidal box, shown in Fig. 306, built of 2 inches  $\times$  10 inches Oregon pine, suitably braced by iron rods and lined with sheet lead called a "pan," in which is a dilute sulphuric acid of 1 to 2 per cent. strength. The solution is kept hot by means of steam passed in through a 2-inch leaden pipe reaching to the bottom of the vat, where it is suitably perforated, while another similar pipe brings in sufficient hot solution to maintain a constant overflow. A baffle plate of lead reaches two-thirds of the way to the bottom of the pan, so that the whole of the pulp has to pass below its lower edge before the sulphide particles can get a chance to rise. The temperature of the solution must be kept above  $60^{\circ}$ , and preferably at about  $80^{\circ}$ , in order to secure good results. The

concentrates with bubbles of gas attached rise through the solution and overflow into a wide launder at one side of the vat, together with a certain amount of fine slime contained in the pulp, while the tailings sink to the bottom and are discharged through an opening in an antimonial metal (90 per cent. Pb, 10 per cent. Sb) seating, closed by a plug of the same material in which is a  $\frac{5}{8}$ -inch hole for ordinary discharge purposes, while the whole plug can be dropped by means of a lever to let out stones, &c., accidentally blocking the opening. These plugs last about 48 hours, during which time the original  $\frac{5}{8}$ -inch orifice becomes enlarged to  $1\frac{1}{2}$  inches. The

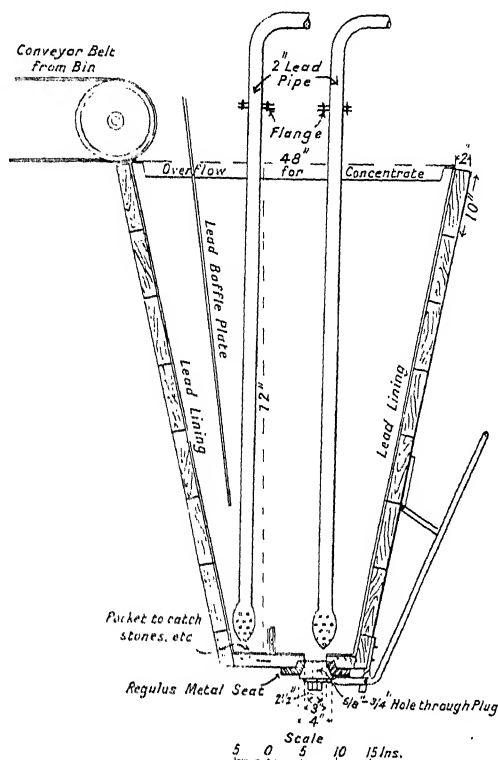


Fig. 306.

\* *E. and M. J.*, Aug. 19, 1905, p. 289; *M. and S. P.*, June 8, 1907, p. 728.

tailings are run into trucks, and the overflowing solution run to a sump. The concentrates (chiefly zinc) overflow from the vats gently, and are dropped suddenly into a plug-bottomed vat, where the gas bubbles become disengaged, and whence the settled concentrates discharge into trucks beneath. The overflowing solution runs through settling tanks, where it deposits any remaining concentrates, together with the slime floated off from the original material treated (the sediment from which tanks is treated on Wilfley tables), and then flows to the sump, where a submerged centrifugal pump made of anti-monial metal raises it back to the solution vats. More recently an air-lift pump has replaced the centrifugal.

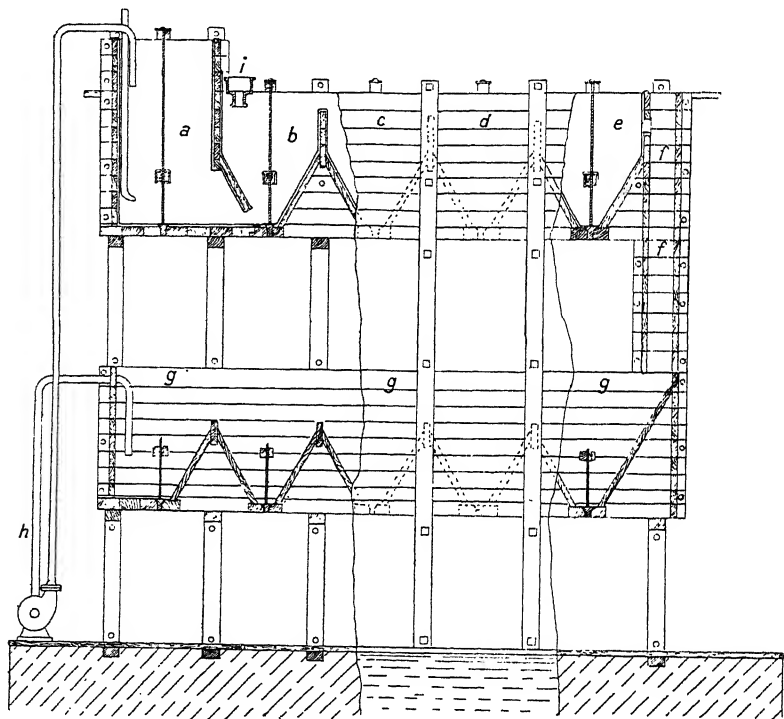


Fig. 307.—Section of Potter Process Vats.

The treating capacity of each vat is about 8 tons per hour. The following figures are an average of six months' results with the process at the Block 14 plant :—

	Per cent. of Total.	Per cent. Pb.	Per cent. Zn.	Ozs. Ag.
Original tailings (from bubbles), . .	..	5.8	15.6	6.6
Concentrates, . . . . .	25	6.1	42.0	11.3
Percentage saving, . . . . .	..	25	67	42

The costs were as follows, per ton of material treated :—Acid, 3s. 3d. ; labour and supervision, 3s. 5d. ; transport of tailings, 1s. 3d. ; coal, 1s. ; water, 6d.—stores, 3d. ; total, 9s. 8d. per ton of original material, or about £2 per ton of concentrates produced.

These costs appear very high, particularly that for acid ; the consumption of acid was 30 lbs. per ton of tailings treated, costing £11 per ton, which could be greatly reduced by manufacturing the acid required on the spot ; the labour cost is also out of proportion to what it would be in other districts or with better arrangements.

The modified Potter process,\* now employed for treating ores of both lead and copper, uses much less acid than the original process, and combines

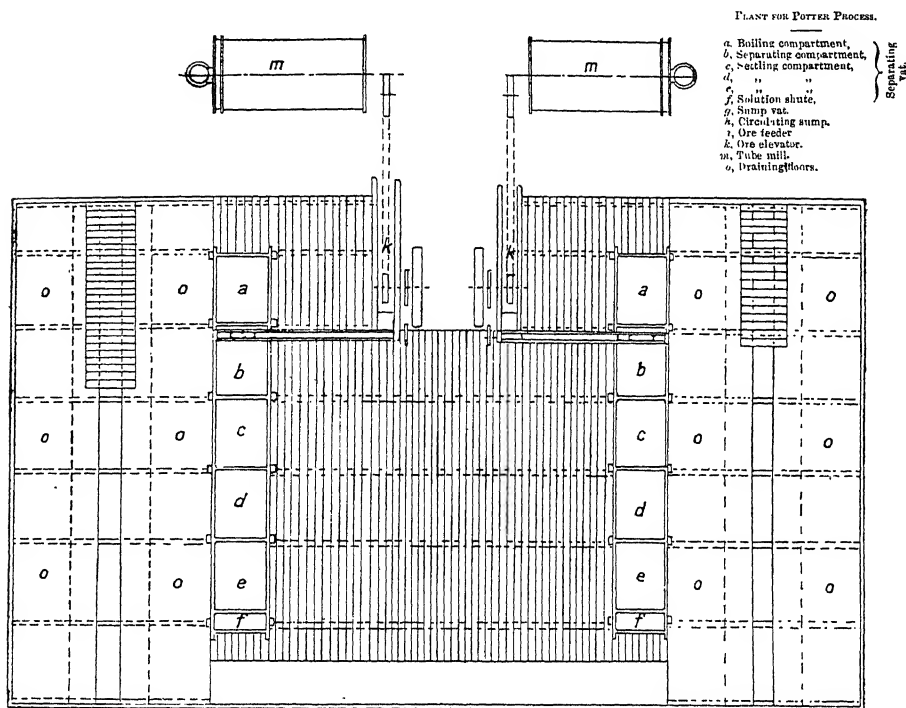


Fig. 308.—Plan of Potter Process Vats.

with the acid a certain amount of oil. In the latest form of the process the acid used for Broken Hill ores is only about 0.6 per cent. or 12 lbs. per ton, and heavy oil is also used in the proportion of 0.5 to 0.8 per cent., equal to from 10 to 16 lbs. per ton of ore.

The latest form of appliance for this process is shown in Figs. 307 and 308. The acidulated solution is circulated continuously through the separating vat *a b c d e f*, and the sump vat *g*, by means of the centrifugal pump *h*, being heated to boiling by injection of steam in the compartment *a* of the separating vat. The oil is mixed dry with the finely pulverised

\* Private communication, T. J. Greenway, May, 1909.



ore ground to 40-mesh size by means of tube- or ball-mills, in any convenient form of dry mixer, and the oiled ore is, by means of the ore-feeding trough *i*, fed continuously into compartment *b*, where it meets with an ascending current of the heated acidulated solution. The gangue settles in compartment *b*, whence it is discharged through a valve into draining trucks, from which it is tipped on to the floors *o*. The floating sulphides are carried continuously by the flowing solution into the settling compartments *c*, *d*, and *e*, where, in consequence of the cooling of the solution and the disengagement of the attached gas, they sink, and are collected and drained in the same manner as the gangue. The solution freed from the flotation products passes down shoot *f* into sump vat *g*, from which, after the deposition of any suspended matter, it is pumped into compartment *a*, to be again used for treating further quantities of ore. The acid consumed is replaced by adding the required quantity to the sump vat.

The first concentrate from Broken Hill ore not being clean, on account of entangling sand tailings, is retreated by the same process, the second concentrate is passed through a continuously fed tube retort to burn off the oil, and is then treated on Wilfley tables, yielding marketable lead and zinc concentrates. The costs of treatment are about as follows:

	s.	d.
Fine grinding, . . . . .	1	6
Labour, steam, and royalty, . . . . .	3	0
Oil, say, 0.5 per cent. at £8. . . . .	0	10
Acid, say, 0.6 per cent. at £6. . . . .	0	8
Retreatment, 50 per cent. at 4s., . . . . .	2	0
De-oiling and vanning, 50 per cent. at 2s., . . . . .	1	0

9s. 6d.

**The Delprat Process.**—This process, never essentially different from the Potter process, became later so like it in detail that considerable litigation ensued between the two companies owning the respective patents, who finally compromised their differences in an agreement to join forces. The original Delprat patent was for the use of a strong acid solution of salt-

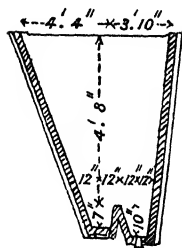


Fig. 309.  
Flotation Vat  
(Delprat Process).

cake, the specific gravity of which would, it was expected, materially assist the bubbles of gas in buoying up the particles of sulphide minerals, and so facilitate a good separation. In practice, however, the use of salt-cake was soon given up, and a certain amount of common salt was added to the dilute sulphuric acid employed in the Potter process, but it has been found that the advantage obtained from the use of salt or salt-cake is by no means sufficient to compensate for the increased cost.

The pyramidal vats employed by the Broken Hill Proprietary Company\* for the Delprat-Potter process were, until recently, 4 feet 4 inches by 3 feet 10 inches at top, and 4 feet 8 inches deep, and differ from those of the original Potter process at the Block 14, chiefly in being provided with a blind pocket, into which stones, bolts, nuts, &c., can fall and be retained without interfering with the operation of the plug, as shown in Fig. 309.

\* *E. and M. J.*, Feb. 16, 1907, p. 321.

These vats were, like those at Block 14, formerly lined with sheet lead, but they are now made simply of wood unlined, and much larger and deeper, 9 feet square by 16 feet deep, the larger size giving better results.

The solution is introduced through iron pipes, and is heated by means of superheated steam at a temperature of 260° F.

The overflow of the vats is to right and left, and the overflows from the whole series of vats collect into one launder, which is led successively into one or other of a series of settling tanks, where the gas bubbles are disengaged and the concentrates settle out, to be subsequently washed with clean water and drained.

The bottom discharge from the vats delivers on to a troughed conveyor belt 3 feet wide set at a slight inclination, which carries off the tailings to be eventually used for filling below ground, while the solution drains back to a sump, from which air-lifts return it to the solution storage tank at the head of the process.

This Broken Hill Proprietary plant is well laid out for labour saving. One man looks after six spitzkasten treating 10 or 12 tons per hour each, and, with the exception of opening and closing valves, &c., everything is moved mechanically.

It is currently reported \* that the Proprietary Company is now using oil in connection with the Potter process, in order to assist flotation, the amount mentioned being about 3½ lbs. per ton of tailings treated.

**The De Bavay Process.**—In this process the ore, first agitated with water and  $\text{CO}_2$  (obtained from chimney gases) in a suitable closed cistern under slight pressure to the condition of a thin paste, is fed, by means of a long feed roller, over a long horizontal lip by the aid of a thin stream of additional water, which distributes it over the surface of a smooth metal table inclined at 35° to the horizontal, and furnished with three staggered rows of tongues, which complete and adjust the distribution. The thin even stream of pulp thus obtained is fed upon the surface of water contained in a small horizontal trough with an overflow lip running its full length, and provided with a separate water supply. Part of the sulphides float upon the surface of the water, which overflows the delivery lip, carrying with it the floating particles by an inclined gutter to a suitable bin. The gangue minerals with the rest of the sulphides that have settled through the water are received upon a slowly moving inclined belt conveyor, which carries up the pulp to a convenient height, and delivers it into a V-

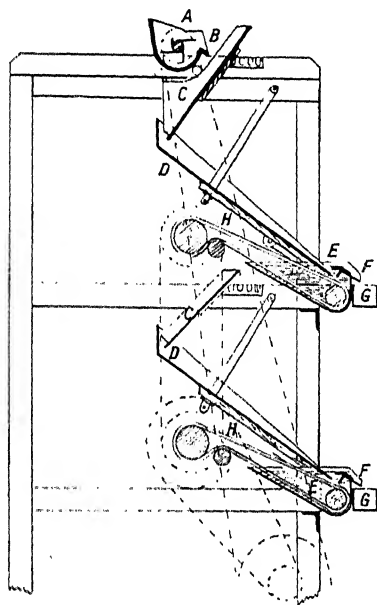


Fig. 310. De Bavay Process Plant.

\* *M. and Sc. Press*, April 11, 1908, p. 494.

launder, where, after the addition of more water, it is again delivered on to an inclined table, and the flotation upon the surface of water repeated; and so on in cascade form as often as may be required to effect a complete separation, and in any case three times.

Fig. 310 \* shows, in diagrammatic section, a portion of the apparatus employed, A being the feed-roller in its feed-box with lip B, C the inclined shoot leading therefrom to the adjustable feeding-table D, E the trough with its lip F and gutter G, and H the conveying belt for "repeating" the unfloated position of the pulp.

The above is the original form of the apparatus introduced at the North Mine, Broken Hill, and careful trials on a small scale indicated the possibility of extracting from carefully selected pulp free from slime and graduated between 40- and 80-mesh nearly all the zinc and 75 per cent. of the lead and silver in the form of a concentrate assaying 44 to 46 per cent. zinc, 11 to 14 per cent. lead, and about 15 ozs. of silver. Although not at liberty to give details, the author understands that the results on a large scale have not quite come up to expectations, and that somewhat extensive alterations in the plant have proved to be necessary. In particular it is stated † that  $\text{CO}_2$  is no longer used, but has been replaced by a solution of  $\text{SO}_2$  made on the ground, and further acidified by an addition of 5 to 10 lbs. of  $\text{H}_2\text{SO}_4$  per ton of ore. Agitation is completed with air under 60 lbs. pressure, and an oil emulsion is added in the agitation vats to assist flotation.

It will be seen, therefore, that all the flotation processes have come round to the use of oil in one form or another. It should further be noted that none of them appear to do much good on true slime, all requiring a fine-grained granular "meal-size" pulp in order to show good results.

The new Elmore vacuum process may be regarded as a combination of the old oil process with the bubble processes, oil being still agitated with the pulp, in order to grease the surface of the metallic sulphide particles, though not to float them afterwards. Flotation is brought about, not by the attachment of bubbles of  $\text{CO}_2$  generated by the action of acid upon carbonates, as in the Potter-Delprat processes, or introduced into the pulp in gaseous form, but most ingeniously by the attachment of bubbles of air and other gases dissolved in the water, which, by the action of a vacuum, are not only liberated, but are caused to expand, thereby greatly increasing their buoyancy. It is found advantageous to add a certain amount of acid to the pulp together with the oil, because, in some hitherto unexplained way, it appears to increase what, for want of a better name, may be termed the "selective action" of the oil upon the metallic particles. The quantity of oil required for the process is much smaller than that employed in the old Elmore process, on the other hand there is no recovery, as practically the total quantity used remains with the concentrates. The total consumption may be from 4 to 12 lbs. per ton of ore treated, while that of acid may vary from nothing up to 8 or 10 lbs. per ton for good results.

Fig. 311, in diagrammatic section, shows the *modus operandi* of the process. The ore is first crushed wet to pass a screen of 20- to 40-mesh by means of stamps, Huntington mills, or other appliances, and the pulp from the mill flows into the mixer, A, into which also, at the point B, small graduated streams of oil and of diluted sulphuric acid are introduced. After agitation

\* E. and M. J., Aug. 25, 1906, p. 344.

† M. and Sc. Press, April 11, 1908, p. 494.

the pulp flows into the open funnel of the feed pipe D, which at first turns down and then vertically upwards for a distance of about 25 feet, forming the short leg of a syphon, of which the tube F is the long leg, having its outlet sealed in the pointed box H; the other long leg E has its outlet sealed in similar fashion in the pointed box G.

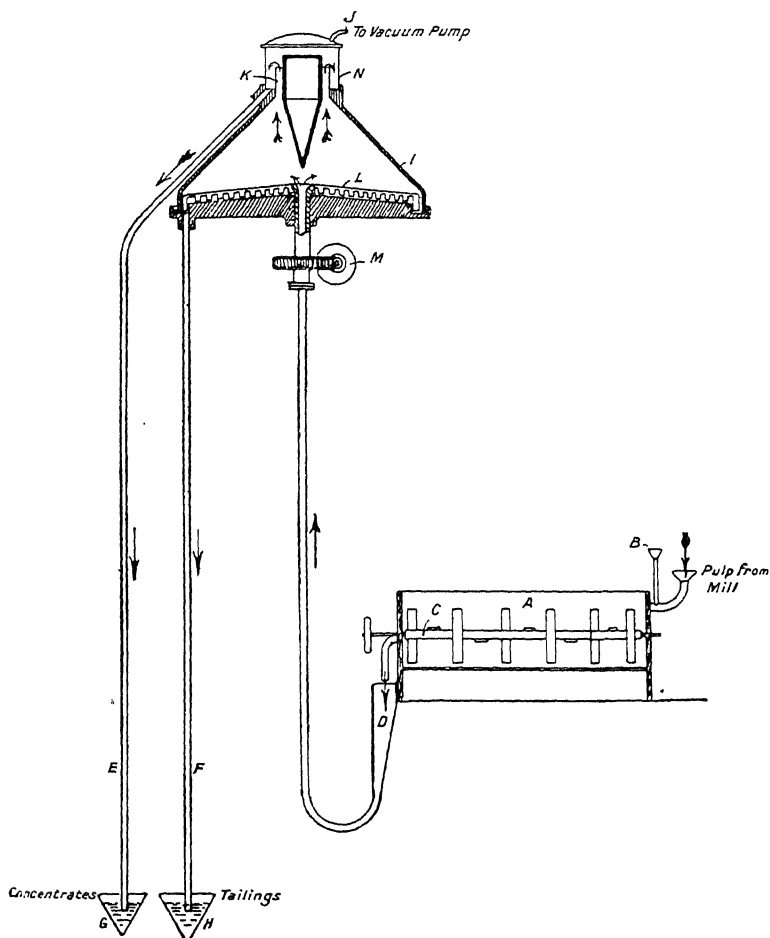


Fig. 311.—Elmore Vacuum Plant (Diagrammatic).

The top of the syphon is formed by the conical separating vessel I, of cast iron and thick glass. To it is attached the pipe J, which communicates with the air pump, and so forms the vacuum. This vacuum is the foundation of the process, as it generates the bubbles and gives the power for circulating the pulp. By the action of the vacuum the pulp, rising in the feed pipe D, fills the separating vessel I, and if the down-flow through the tailings-pipe F be slightly throttled a small quantity of the water is caused to overflow

the lip of the annular space at the top of the chamber, carrying with it the thick scum of oiled concentrates. This scum, buoyed upwards by the continually expanding air-bubbles, collects as a spongy mass at the very top of the apparatus. The concentrates that overflow the lip are carried down the pipe E by the water, and settle in the pointed box G, from which, at intervals, they are discharged into a centrifugal separator, and there dried. The sandy gangue matter of the pulp settles on the flat bottom of the chamber I, where it is stirred, in order to disengage any particles of concentrate mechanically entangled, and at the same time gradually worked towards the circumference with its (generally multiple) tailings-pipe exits by means of the angled rake-blades, which are caused to revolve by the worm and wheel at M. The annular space at K is surrounded, in the small machines, by a glass cylinder, while in the large ones it is provided with heavy plate glass windows, so that the discharge of the concentrates over the lip may be observed and controlled. It is obvious that, so long as the vacuum is maintained and a constant stream of pulp is supplied to the mixer, a continuous and quite automatic separation and discharge of concentrates and tailings will be secured.

The conical separating chamber is usually made 5 feet in diameter, and the power required for operating such a unit of plant treating 35 to 45 tons of ore per twenty-four hours, apart of course from that required for crushing, is usually not over  $2\frac{1}{2}$  to 3 H.P.

The percentage of extraction varies a good deal, but is always fairly high. It is adversely affected by the presence of a considerable quantity of calcite in the gangue, which prevents the maintenance of an acid solution; whereas, when treating ores free from calcite, and particularly when using acid mine waters, the recovery is very good.

At *Tywarnhaile* (Cornwall), when treating low-grade pyritic waste dumps, the average pulp assay of which was only 0.41 per cent., a concentrate was obtained of 8.18 per cent. copper and tailings of 0.13 per cent., showing a saving, even on this very poor material, of about 68 per cent.

At *Dolcoath* an ore containing 2.41 per cent. copper yielded concentrates with 17.4 per cent. copper and tailings with only 0.23 per cent. copper, showing a saving of 92 per cent.

At the works of the *Zinc Corporation, Ltd.* (Broken Hill)\* zinc-bearing "tailings" from jig-concentration mills are crushed by means of wet-grinding pans through a 30-mesh screen, and the pulp passed through machines of the general design of that shown in Fig. 311. The plant is arranged in units, of which there are sixteen, each consisting of one wet-grinding pan, agitator, mixer, and vacuum plant, and the capacity of each unit is from 40 to 45 tons per day. The sulphuric acid required is manufactured on the spot by the Carmichael-Bradford process,† and the consumption of it is from 10 to 20 lbs. per ton of material treated; the oil used is Texas fuel oil, and the consumption is from 6 to 8 lbs. per ton. The recovery of lead and zinc sulphides by the vacuum process is good, the tailings (consisting chiefly of garnet, rhodonite, and feldspar, with some quartz) being remarkably poor. The mixed concentrate at first obtained, however, proved saleable only at a low figure, and accordingly it is now separated into two products by passing it rapidly through a furnace, in order to flash off the oil, and then stirring up with water and running the

\* *Min. and Scientific Press*, March 13, 1909, p. 392.

† *q.v.*, Chapter vi.

pulp over Wilfley tables. There is a certain amount of loss, due partly to oxidation in the furnace and partly to the production of about 3 per cent. of slime which is lost on the Wilfleys, but the improved prices realised for the separated products much more than compensate for the loss. The following figures show the average composition of the original pulp and of the products :—

	Zinc.	Lead.	Silver.
Original pulp, . . . . .	20 %	5.75%	8 ozs. per ton.
Vacuum concentrate, . . . . .	43 %	11 %	17 „
Lead concentrate from Wilfleys, . . . . .	15 %	58 %	39 „
Zinc „ „ . . . . .	46.5%	7.25%	16 „

The total recovery is said to be 90 per cent. of the zinc, 73 per cent. of the lead, and 85 per cent. of the silver.

The costs of working are said to be—

	s.	d.
Grinding, . . . . .	1	3.96
Vacuum process, . . . . .	2	4.00
Drying and de-oiling, . . . . .	1	3.64
Wilfleys, . . . . .	0	7.48
	5s. 7.08d.	

**The Macquisten Process.\***—This process does not use any oil, air- or gas-bubbles, and does not even require the ore to be first dried. It simply depends upon presenting the particles of mineral to a surface of water at a suitable angle a sufficient number of times.

The essential part of the apparatus employed is a cast-iron cylinder about 6 feet long  $\times$  1 foot in diameter, supported horizontally by means of rings on four small rollers, and weighing about 450 lbs. The discharge end of the tube is open, but extends into a separating box, by means of which and of a water-tight joint the level of the water in the tube is kept about 3 inches above the bottom. The inlet end of the tube is closed, except for a hole large enough to admit the pulp-feed pipe. Inside the tube is cast a series of helical ridges,  $\frac{3}{8}$  inch wide and  $\frac{1}{4}$  inch high, and  $1\frac{1}{2}$  inches apart, which thus form a series of flat grooves in which, as the tube revolves, the particles of ore and gangue are raised and rolled or slid back upon the surface of the water in the lower part of the tube. In practice the tube is revolved at about 30 revolutions per minute in the direction of the helix. Some particles of sulphides are so comparatively difficult to float that even after passing through four such tubes in series they remain with the gangue and refuse to float, but the vast majority are floated after passage through only two tubes in series.

The separating-box has an overflow directly opposite the mouth of the tube, over which water flows in a layer only  $\frac{1}{32}$  inch deep, carrying the floated sulphides to a concentrate tank, and a bottom discharge which takes the gangue either to the waste launder or to a second or further series of tubes. Each tube has a capacity of about 5 tons of ore per twenty-four hours.

At the *Adelaide Reduction Works, Golconda, Nev.*, chalcocopyrite ores

\* *E. and M. J.*, Oct. 26, 1907, p. 765.

containing 2.2 per cent. copper are crushed by means of stone-breaker and rolls to  $\frac{1}{4}$ -inch to  $\frac{3}{8}$ -inch size, and then by means of 5 feet Huntington mills to 10-mesh size. The delivery from the Huntingtons goes to spitzkasten, which eliminate slimes, and the underflow from them is passed over 30-mesh Callow belt screens, the over-size being re-crushed in Huntington mills. Dewatering tanks deliver the pulp to a series of 25 tubes in parallel, from which it passes successively through three similar series of tubes. The concentrates from all the tubes flow through launders to settling tanks at either side of the mill, and the water from these, as well as from the tailings' dumps, is returned by means of centrifugal pumps for re-use. It becomes more efficient by repeated use, but whether the increase in surface tension is due to the formation of a small quantity of free acid or to a gradual increase in mineral salts dissolved has not yet been ascertained. The amount of power required for the tubes is very much less than for jigs and tables. Ore containing 2.2 per cent. copper is concentrated in the ratio of 11 to 1, yielding a concentrate assaying 22 per cent. of copper, and tailings of only 0.2 per cent. copper. This is equivalent to an extraction of about 90 per cent. on

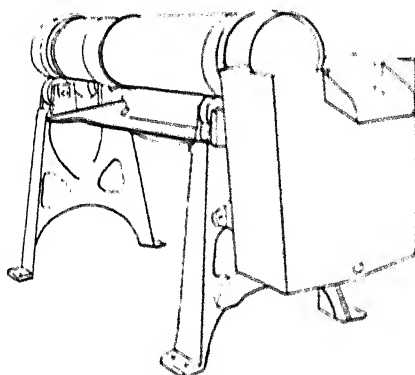


Fig. 312.—Macquisten Tube.

the sand or deslimed portion of the pulp. As in all the flotation processes fine slime seems to be not adapted to treatment, and as the amount of it is about 30 per cent. by weight of the original ore, and its assay value is about equal to that of the original ore, the actual extraction by the process is only  $70 \times 90 \div 100 = 63$  per cent. of the copper contents of the original ore. Fig. 312 shows a single tube and separating box.

This simplest of all flotation processes appears to have a possible field of usefulness in connection with separations of unaltered sulphide minerals from gangue, but it seems to be very doubtful if the separation would be as complete in the case of a sulphide like blende as it might be, for example, with pyrites or chalcopyrite.

**The Murex Magnetic Process.**—This process is mentioned here because it is a combination of the magnetic and flotation processes. A generalised view of the plant is shown in Figs. 313 and 314. The essential feature of the process is the thorough incorporation with the pulp of a certain small quantity of oil partially charged but not completely loaded with finely

Fig. 313.

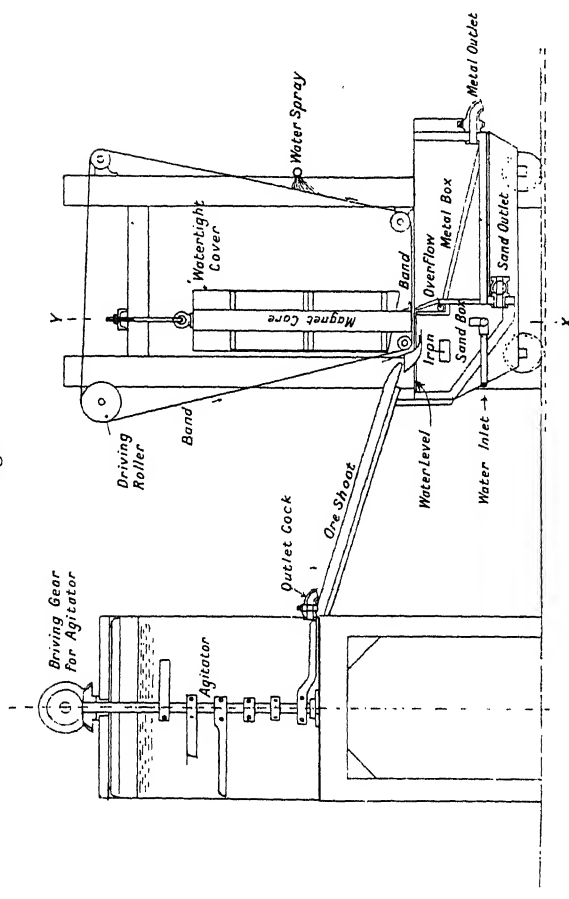
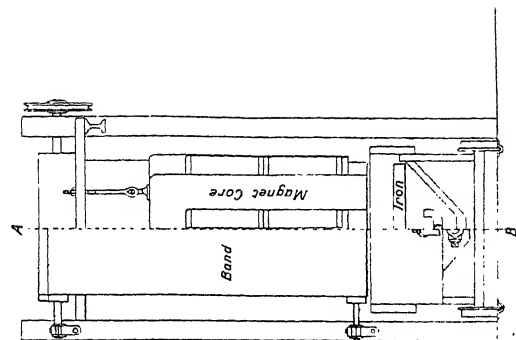


Fig. 314.



Figs. 313 and 314.—Plant for Murex Magnetic Process (Diagrammatic).



ground magnetite. The result of thoroughly incorporating with the pulp this so-called magnetic "fluid" is to coat the sulphide particles with a thin film of oil and magnetite, or rather perhaps to cause them to form, under the influence of surface tension, composite aggregates with the particles of magnetite. As soon as they come under the attraction of the magnet under which the pulp is fed by the curved shoot shown, these behave as magnetic particles, and are carried, by the travelling belt shown, over the partition separating the sand from the metal tanks, in the latter of which they subside, such of them as remain clinging to the belt being washed off by a fine water spray above.

Tests on Broken Hill ores from the British Mine that by ordinary wet concentration methods yield tailings still containing 5.1 per cent. lead, 15.7 per cent. zinc, and 5.9 ozs. of silver per ton showed that the Murex process left in the tailings only 1.5 per cent. lead, 2.1 per cent. zinc, and 1.95 ozs. of silver per ton.

The separation of one sulphide mineral from another is said to be effected to a degree quite commercially successful by means of additions of certain salts to the emulsion of magnetite and oil, sulphate of alumina being one of the salts that is said to give marvellous results.

Hitherto the process may be said to be still in the experimental stage, but it is probable that it will be seen at work on a commercial scale before long.

As in the case of all other mechanical separation processes, both wet and dry, the greatest bugbear is the presence of fine slime in the pulp. Although a close saving and clean tailings are shown when treating fairly evenly-sized fine sand and meal particles, the recovery on fine slimes appears to be no better than that shown by most other processes.

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